

BASIC MINE VENTILATION



A—PROJECTING MOUTH OF CONDUIT. B—PLANKS FIXED TO THE MOUTH OF THE CONDUIT
TO STOP THE WIND FROM BLOWING IN

"I have omitted all those things which I have not myself seen, or have not read or heard of from persons upon whom I can rely. That which I have neither seen, nor carefully considered after reading or hearing of, I have not written about. The same rule must be understood with regard to all of my instruction, whether I enjoin things which ought to be done, or describe things which are usual, or condemn things which are done".

Georgius Agricola, Preface to De Re Metallica, 1556.

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INTRODUCTION

Safe, sustainable underground mining of any scale is not possible without an effective ventilation system. Ventilation is required in the underground workings to:

- Dilute gaseous and particulate pollutants to concentrations which are not injurious to the health and safety of the workforce
- Maintain thermal comfort through the provision of adequate air velocity
- Assist with maintaining workforce morale and productivity through the provision of high quality underground environmental conditions

An efficient and effective mine ventilation system is not achieved by accident. It requires incorporation of a solid understanding of scientific ventilation principles into the very core of the mine planning and management process.

Important lessons can be learnt from history which has emphasised time and time again is that the knowledge required to implement effective mine ventilation has often been significantly more advanced than the practices adopted in the mines at the time. The terrible loss of life in the British coal industry in the 19th Century (which did not abate until the belated introduction of legislation early in the 20th Century) is a sobering example. More recent examples of the same phenomena in Australia include the many cases of mesothelioma contracted by underground miners in Wittenoom in the 1950's and '60's as well as the Moura mine disasters of the 1980's and 1990's. In all of these examples, the health and safety risks were well known, as were the engineering requirements necessary in order to minimise these risks. The costs of installing and managing effective ventilation systems must have been minuscule compared with the subsequent outlays for compensation. One inference which could be drawn from all of this is that ignorance and indifference have been (and perhaps still are) important factors.

Brief History of Mine Ventilation

Pre Middle Ages	There is evidence to suggest that the need to establish ventilation circuits with intake and return airways was understood and practiced by the Greeks. For example there was twin access development and divided shafts in the Laurium silver mines of Greece (600 BC). In the Roman times, slaves waving palm fronds were used to promote air circulation. (Pliny AD 23-79). More generally, air movement was created by up-draughts of warm air from fires lit for this purpose underground.
Middle Ages Industrial Revolution	Mine ventilation practices of the day are illustrated and described in "De Re Metallica", by Georgius Agricola, 1556. This was amongst the first books ever printed and remained a standard mining text for the next 200 years. The book describes horse and human powered centrifugal fans; bellows connected to wooden conduits, ventilation doors and shaft collar deflectors, which divert winds into the mine workings. Agricola also described the dangers of "blackdamp" (oxygen reduced air) and the explosive "firedamp" (an air and methane mixture "likened to the fiery blast of a dragon's breath").

Industrial Revolution – Information Age	This period saw an unprecedented increase in demand for mineral commodities including coal, metalliferous ores and industrial minerals. Substantial growth in mining activity, (much of it underground) was required to satisfy this new demand. Larger scale mining led to increasing numbers of miners suffering from health problems contributed to by poor ventilation, including “black lung”, mine explosions and fires. Hot working conditions at depth due to ventilation limitations also restricted the extent of mining activities. The increasing slaughter of men, women and children in British coal mines as a result of explosions and fires eventually served as a major impetus to drive the development of the science of underground ventilation in the 19th Century. John Atkinson, a British mining agent presented his famous paper “On the Theory of the Ventilation of Mines” in 1854 and Atkinson’s Equation still forms the basis for all mine ventilation engineering. Interestingly, whilst the importance of Atkinson’s paper and theories were recognised at the time they were published, they received little interest and were considered too much for the engineers of the day. It wasn’t until some 60 years after Atkinson’s death that his theories were “rediscovered”. In the 1920’s and 1930’s large axial fans based originally on aeroplane propeller designs began to become popular. Analogue computer models were used in the 1950’s to analyse ventilation networks. The introduction of diesel equipment underground in the 1960’s and 1970’s required substantial increases in mine ventilation capacity. Many mines of the day embarked on massive upgrades to ventilation shafts and main surface fans. Great advances in large fan design were made in the 1960’s and 1970’s. At about the same time, improvements in large scale refrigeration technology began to allow very deep orebodies to be accessed, particularly in South Africa.
Information Age	The most notable feature has been the development of computerised mine ventilation network analysis. This has made prediction of fan requirements and airflow distributions in complex mine ventilation circuits feasible using desktop computers.

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“All substances are poisons; it is only the dose that separates the poison from the remedy”

Paracelsus - 15th Century

1 CONTAMINANTS AND EXPOSURE STANDARDS

The basic requirement for the mine ventilation system is to provide air for people to breathe and in a state that will not cause any immediate or future ill effects.

Because of the processes of mining, if positive airflow through the workings was not provided the air would very quickly become stale, contaminated and unfit for human consumption. The ventilation system must therefore be sufficient to deal with the contaminants released during mining. If they are not adequately dealt with, as they are identified, they may become at best a discomfort to mine workers, and at worst cause serious or even fatal illness.

The prime contaminants produced during mining are

1. dust
2. heat
3. gases (including water vapour i.e. Humidity)

and the prime method for dealing with these is an effective ventilation system that

1. supplies oxygen and cools
2. dilutes dust and gases and
3. removes the contaminants from the workplace.

It is possible that any known substance will be identified, in either the air we breathe or the food we eat. Although the human body is equipped to reject or absorb these substances this can only be done providing the quantities involved are not excessive.

The search to identify specific substances and their harmful concentrations is on going with limits constantly under review. Levels of atmospheric contaminants that are “safe” in an occupational health and safety environment are often difficult to determine. There are many factors which must be considered including:

- Variability in response of individuals to contaminants
- Synergistic effects (i.e. combined effect of simultaneous exposure to several contaminants)
- Work rate (affects respiration rate)
- Work cycle (“compressed” work cycles give the body less recuperation time between exposures)
- Changes in scientific understanding

As a result of the above factors, recommended maximum exposures to various contaminants are constantly under review and the latest information should always be sought

Worksafe Australia produce a publication titled “Exposure Standards for Atmospheric Contaminants in the Occupational Environment” which provides comprehensive information on the present exposure standards for known hazardous substances along with information of substances presently under review. This standard represents the concentration of substances that should not cause undue discomfort or impair the health of persons when exposed to these

substances. They do not however represent a “no-effect” level to all persons exposed to the concentrations stated. They are of course based on current knowledge and research and are subject to change. This information can be found on the Internet at www.nohsc.gov.au

This website tells us that when interpreting the actual “Exposure Standard” it is important to note that they*“have been established on the premise of an eight-hour exposure, during work of normal intensity, under normal climatic conditions and where there is a sixteen-hour period between shifts to permit elimination of any absorbed contaminants.”*

And “*Heavy or strenuous work increases lung ventilation, thereby increasing the uptake of airborne contaminants. Similarly, heavy physical work under adverse climatic conditions, such as excessive humidity or heat, or work at high altitudes, may lead to an increased uptake of contaminants. It is therefore of particular importance that any evaluation of the working environment considers the lung ventilation rate where there is a significant airborne concentration of contaminant.*”

This website also provides descriptions of the specific exposure standards for peak, short term, time weighted average and general excursions.

1.1 Peak Limitation

For some rapidly acting substances and irritants, the averaging of the airborne concentration over an eight-hour period is inappropriate. These substances may induce acute effects after relatively brief exposure to high concentrations and so the exposure standard for these substances represents a maximum or peak concentration to which workers may be exposed. Although it is recognised that there are analytical limitations to the measurement of some substances, compliance with these ‘peak limitation’ exposure standards should be determined over the shortest analytically practicable period of time, but **under no circumstances should a single determination exceed 15 minutes.**

1.2 Short Term Exposure Limit (STEL)

Some substances can cause intolerable irritation or other acute effects upon brief overexposure, although the primary toxic effects may be due to long term exposure through accumulation of substances in the body or through gradual health impairment with repeated exposures. Under these circumstances, exposure should be controlled to avoid both acute and chronic health effects.

Short-term exposure limits (STELs) provide guidelines for the control of short-term exposure. These are important supplements to the eight-hour TWA exposure standards that are more concerned with the total intake over long periods of time. Generally, STELs are established to minimise the risk of the occurrence in nearly all workers of:

- intolerable irritation
- chronic or irreversible tissue change, and
- narcosis to an extent that could precipitate industrial accidents.

STELs are expressed as airborne concentrations of substances, averaged over a period of 15 minutes. This short term TWA concentration should not be exceeded at any time during a normal eight-hour working day. Workers should not be exposed at the STEL concentration continuously for longer than 15 minutes, or for more than four such periods per working day. A minimum of 60 minutes should be allowed between successive exposures at the STEL concentration.

1.3 Time-Weighted Average (TWA)

Except for short-term exposure limits, or where a peak value has been assigned, the exposure standards for airborne contaminants are expressed as a time-weighted average (TWA) concentration of that substance over an eight-hour working day, for a five-day working week. During periods of continuous daily exposure to an airborne contaminant, these TWA exposures permit excursions above the exposure standard provided they are compensated for by equivalent excursions below the standard during the working day. However, it is not necessarily acceptable to expose workers to concentrations significantly higher than the exposure standard solely because the exposure is for less than an eight-hour day or because the exposure occurs only occasionally. Permissible variations in the exposure standard for such situations are dependent on such factors as the acute effects of short-term exposures, or on the relationship between accumulation and elimination of the body burden of the material or its metabolites, and should only be accepted in the light of expert advice.

1.4 Guidance on General Excursion

In practice, the actual concentration of an airborne contaminant arising from a particular industrial process may fluctuate widely with time, with some of the major excursions giving rise to a significant proportion of the overall exposure.

Even where the TWA exposure standard is not exceeded, there should be some control of concentration excursions. A practical approach to control has been developed, based on observations of the variability in concentrations observed in industrial environments. A process is not considered to be under reasonable control if short term exposures exceed three times the TWA exposure standard for more than a total of 30 minutes per eight-hour working day, or if a single short term value exceeds five times the TWA exposure standard. It should be emphasised that guidance of this type, aimed at placing some restraint on concentration excursions, is not directly health-based and does not supersede any STEL or peak limitation set.

Where adequate toxicological or epidemiological data allows the assignment of a STEL, the STEL will supersede this guidance on general excursion.

1.5 Adjusting the Eight-hour Exposure Standard for Longer Periods

“Compressed” work cycles (usually in fly-in/ fly out operations) consisting of 12-hour shifts for up to (and sometimes exceeding) 14 days duration is now commonplace. As yet, there is no agreement on the extent to which TWA exposure standards should be reduced in response to various work patterns. Specialist consideration and expert advice should be sought in the specification of modified exposure standards.

NOHSC recommends the use of the Brief and Scala (1975)¹ model to adjust the time-weighted average (TWA). This method was chosen because it is a simple calculation, it is the most conservative model developed and does not require any detailed knowledge of the substance.

$$\text{Adjusted (TWA) Exposure Standard} = \frac{8 \times (24 - h) \times \text{Eight Hour Exposure Standard}}{16 \times h}$$

Where h = hours worked per day

¹ BRIEF, R., SCALA, R., “Occupational Exposure Limits for Novel Work Schedules.” American Industrial Hygiene Association Journal. 36: pp467-469, 1975.

Example

The eight-hour exposure standard for Carbon Monoxide is 30 ppm. The adjusted TWA exposure for Carbon Monoxide (CO) assuming a twelve-hour working shift is calculated as follows.

$$\text{Adjusted TWA} = \frac{8 \times (24 - 12) \times 30}{16 \times 12} = 15 \text{ ppm}$$

No adjustment is necessary for the excursion limits of peak concentrations or the short-term exposure limit (STEL).

A Comparison of some Atmospheric Exposure Standards Adjusted for Shift Duration

CONTAMINANT	TWA			STEL Any
	8-hour	10-hour	12-hour	
* Carbon Monoxide (CO)	30	21	15	400
* Carbon Dioxide (CO ₂)	5,000	3500	2,500	35,000
* Nitrogen Dioxide (NO ₂)	3	2	1.5	5
* Nitrous Oxide (NO)	25	18	12	UR
* Sulphur Dioxide (SO ₂)	2	1	1	5
* Hydrogen Sulphide (H ₂ S)	10	7	5	15
# Respirable Dust	5.00	3.50	2.50	NA
# Quartz Bearing Dust	0.20	0.14	0.10	NA
# Respirable Combustible Dusts	2.00	1.40	1.00	NA

NOTES: * = ppm # = mg/m³ UR = Under Review NA = Not Applicable

“Some mines are so dry that they are entirely devoid of water and this dryness causes the workmen even greater harm, for the dust, which is stirred up and beaten up by digging penetrates the wind-pipe and lungs and produces difficulty in breathing and the disease the Greeks call asthma. If the dust has corrosive qualities, it eats away the lungs, and implants consumption in the body. In the mines of Carpathian Mountains women are found to have married seven husbands, all of whom this terrible consumption has carried off to a premature death.”

Georgius Agricola - “De Re Metallica” (1556)

2 OCCUPATIONAL HEALTH AND SAFETY

There is a surprisingly broad range of environmental hazards in underground mining, including poisonous, asphyxiant, carcinogenic or explosive dusts and gasses and extremes of heat and humidity. A significant number of these hazards can result in serious health problems, ranging from long-term physical impairment (e.g. lung disease) to immediate death (e.g. carbon monoxide poisoning, or heat stroke). It is the role of a mine's ventilation systems to control these hazards, but before any ventilation engineering design work is carried out, it is vital that the hazards are well understood. This chapter describes some of the more common mine environmental hazards.

It should be noted that the human body is very resilient and well equipped to deal with nature's climatic and atmospheric extremes. However, some processes involved in activities such as mining produce contaminant levels and environmental conditions which are beyond those with which the body's natural defence systems can cope.

The human capacity to function at the maximum potential will reduce rapidly in atmospheres contaminated with dust, gases, heat and humidity and the long term effects of exposure to contaminants may have serious impact on general medical health.

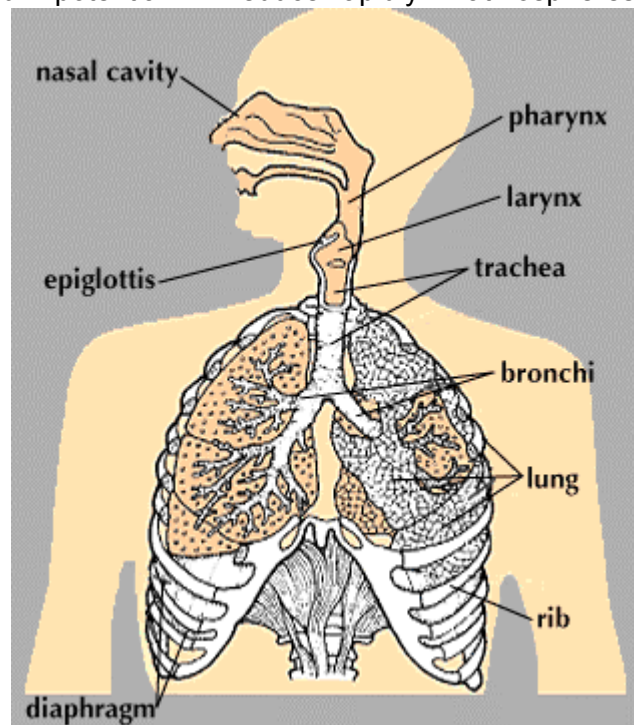
2.1 The Respiration System

Medical science has proven the human body to be very resilient and well equipped to deal with nature's climatic and atmospheric extremes. However it has also shown that some processes involved in today's manufacturing and industry may have a detrimental effect on parts within the human body.

Gases, vapours and dusts may enter the human body in three ways

- Inhaled into the respiratory system
- Ingested with food and saliva into the digestive system
- Absorbed through the skin

However it is the respiratory system which provides the major mode of entry into the body

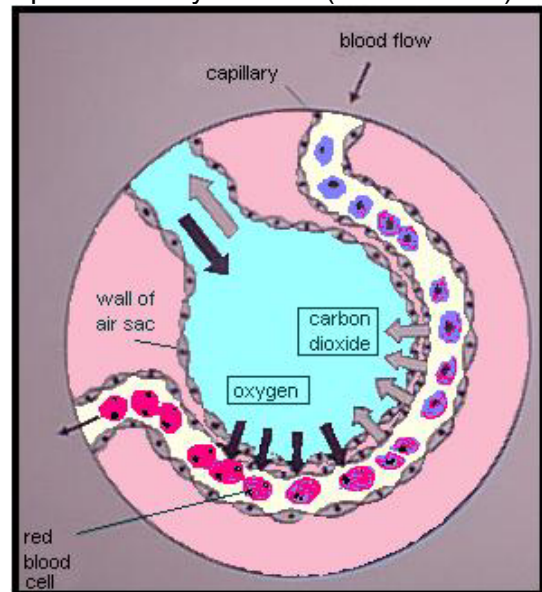


The respiration system is the main point of entry into the body for many of the particulate and gaseous contaminants found in the underground environment. It is therefore important to have a basic understanding of the principles behind the operation of the human respiratory system.

Every organ in the body requires oxygen. Oxygen is 'captured' in tiny air sacs (called alveoli) in the lungs. When air is breathed in, it passes through the walls of the alveoli and surrounding capillaries into the blood stream. The blood transports the oxygen to the body tissues, where it is consumed in the process of energy production, producing carbon dioxide as a waste product. The blood stream carries the carbon dioxide back to the air sacs to be breathed out. The difference between oxygen and carbon dioxide concentrations in blood in the capillaries and the air in the alveoli causes an exchange of gases.

The respiratory system is basically a ventilation system, which supplies air to the alveoli.

By the very nature of their function, the lungs are exposed to any dusts, fumes, smoke, aerosols, mists, gases or vapours small enough to remain airborne.

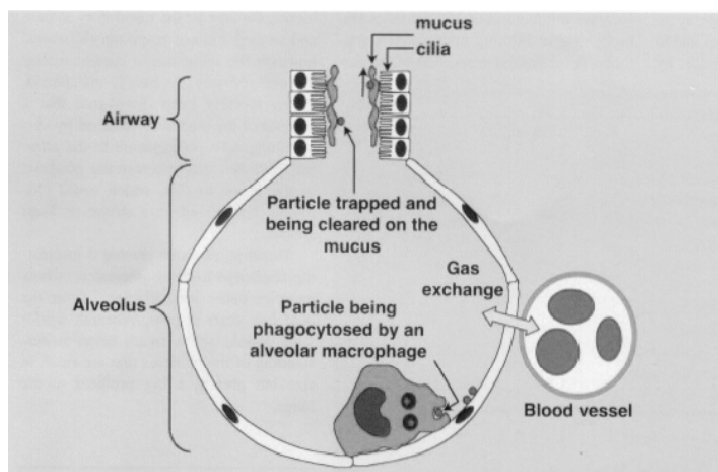


An adult breathes between 2.0 litres and 4.0 litres of air per minute (l/m) and during times of hard work, this can increase to about 8.0 l/min. Because more air is inhaled as work becomes harder, it is easy to understand why workers in heavy dusty jobs such as mining and construction are more likely to suffer from dust and other contaminant-related lung disorders.

By the very nature of their function, the lungs are exposed to any dust, fumes, smoke, aerosol, mist, gas or vapour that is small enough to remain airborne.

The human respiratory system has a number of defence systems which are designed to deal with the concentrations and size ranges of dust normally found in nature. These include:

- **Nasal hairs.** These act as pre-filters. When we inhale, the mucus coated membranes of the nasal passages and hairs of the nose aid in capturing almost all of the coarser particles.
- **Cilia.** The trachea and bronchi are lined with sticky mucus that is wafted up these airways via the action of cilia, which are hair-like cells acting with a wave-like motion. The dust is moved upwards to the mouth, where it is either expectorated or swallowed. Inhalation of very coarse dust particles causes coughing and sneezing, which has the effect of speeding up the removal process. The finer dust particles will however reach the alveoli.
- **Macrophages.** These are relatively large white cells (up to 10 μm diameter), which lie on the surfaces of the alveoli. The cells are mobile and their role is to completely engulf the fine foreign particles which enter the alveoli. Once this has occurred, the macrophages



usually then move up the respiratory system and are eventually expectorated or swallowed. Certain dusts are toxic to macrophages (including free crystalline silica and asbestos) and these substances cannot therefore be expelled from the lungs. They remain in the lung tissue, resulting in a gradual incapacitation of the lungs.

2.2 Dust

In nature dust is mainly formed by the grinding action of wind blown particles on rocks, the impact action of rocks on each other as they knock together during landslides, and the tumbling action in rivers and ocean wave motion. Because this type of impact is limited, dust formed in this way is rather coarse e.g., beach sands, muds and clays.

In mining dust is formed by powerful concentrated forces such as blasting, drilling, crushing, and grinding and consequently forms much smaller dust particles than those formed by nature.

Dust is a result of the disintegration of matter and the size of the dust particle produced is determined by the impact per unit area. For example striking a rock with a hammer will split the rock into large pieces forming coarse dust particles. If we were to use the same force using a chisel it would break only a small piece of the rock into fine dust particles because the force is directed onto a much smaller area.

As an example of dust production, crushing 1mm^3 of rock to $1\mu\text{m}$ particle sizes would yield 1,000 million dust particles. In a drill hole 3.6m deep and 32mm diameter the volume of rock removed is $2,895,291\text{ mm}^3$ producing 2.89×10^{15} dust particles of $1\mu\text{m}$ diameter. If it takes 10 minutes to bore the hole and there is a ventilating airflow of $20\text{m}^3/\text{s}$ each mm^3 of air would be contaminated with 241 particles. If the particles had a density of $3\text{kg}/\text{m}^3$ the eight-hour exposure level would be $0.7\text{ mg}/\text{m}^3$. In reality the majority (99%) of drilling particles are much larger than $1\mu\text{m}$ and are contained in the drilling water.

Some of the more significant sources of dust in underground mines include blasting, movement of rock in stopes, mucking operations, mechanical rock cutting (e.g. raise drilling, road-header etc), ore passes, rock breakers, crushers and conveyor transfer points. Dust is also liberated to the ventilating air from by the tyres of passing traffic lifting the dust from the surface of the mine roadway.

2.2.1 The Hazard of Dust

The long-term effect of dust inhalation is not necessarily confined to the respiratory system and there is some research that suggests that the respiratory system merely provide the mode of entry into the body, other modes being absorption and ingestion. The respiratory system is very selective with respect to the size and quantity of dust retained in the lungs. It is not simply a matter of the quantity of dust in the atmosphere that dictates the amount deposited in the lungs, it is also the duration of exposure and the rate of deposition to the critical area.

The size of dust particles is measured and expressed in microns (μm). The smallest dust particle that can be seen with the naked eye under good conditions (i.e. black on white with good lighting) is around $25\mu\text{m}$ and It is generally accepted that the smallest particle visible to the naked eye can be as large as $50\mu\text{m}$.

Dust particles in mine airflow will settle out of laminar airflows according to Stokes Law. If the airflow is turbulent then the motion of airborne particles is unpredictable are more likely to be removed from the airflow by coagulation and impingement rather than settling.

2.2.1.1 Settling

The terminal velocity of an airborne dust particle is dependant upon the atmospheric drag force holding the particle up and the gravitational force pulling it down. This is known as Stokes Law.



Where

G = Gravitational Force

d = Geometric diameter of the sphere (m)

w_s = Density of the sphere (kg/m^3)

w_a = Density of the air (kg/m^3)

g = Acceleration due to gravity (m/s^2)

F = Drag Force

v = Velocity of the particle (m/s)

η = Viscosity of the fluid (kg/m.s)

Stokes Law

Stokes Law is applied to spheres

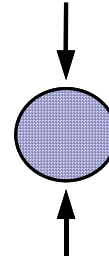
$$G = F$$

$$\frac{1}{6} \pi d^3 (w_s - w_a) g = 3 \pi d \eta v \quad \text{Equation 2-1 Stokes Law}$$

Therefore
$$v = \frac{d^2 g}{18 \eta} (w_s - w_a)$$

Gravitational Force

$$\frac{1}{6} \pi d^3 (w_s - w_a) g$$



Drag Force

$$3 \pi d \eta v$$

Brownian Motion

The Brownian motion is a random motion that occurs when dust particles collide with gas molecules in the air with no net tendency to move downwards. This motion completely masks observations made on the particle in question and the resistance to the particle becomes less and they tend to “slip” past the gas molecule at a speed faster than that indicated by Stokes law.

Slip Corrected Velocity

Cunningham (1910) introduced a slip correction

$$v_c = v \left(1 + 2A \frac{\lambda}{d_p} \right) \quad \text{Equation 2-2 Cunningham's slip corrected velocity}$$

Where

V_c = Slip corrected velocity of the particle (m/s)

A = a constant which varies from 0.7 to 0.9

λ = the mean free path of the gas molecule ($6.53 \times 10^{-8} \text{m}$ at 20°C and 101.3kPa)

d_p = diameter of the falling particle (m)

Particles having terminal settling velocities of the same order as the displacement caused by the Brownian motion will remain suspended, even in still air.

The terminal settling velocities may be calculated from the above equations. For example a quartz particle with a geometric diameter of $10 \mu\text{m}$ will have a slip corrected settling velocity of $8.04 \times 10^{-3} \text{m/s}$ and if released from a height of 4.0m would require 8.29mins to settle to the ground and $1 \mu\text{m}$ particle would take 12.8hours . In fact because of the turbulence associated with mine ventilation along with the Brownian motion it would be considerably greater.

In those cases where the settling velocity is equal to or less than the Brownian displacement the particle will never settle, even in still air.

Physical Properties of Airborne α -Quartz Particles at 19.0/20.0 °C and 85kPa

Equivalent geometric diameter of Quartz particle (μm)	Aerodynamic Diameter (= diameter of unit density sphere) (μm)	Stokes diameter (from experimental data $d_p/d_s = 1:1.67$) (μm)	Stokes Terminal Velocity (m/s)	Slip-corrected terminal velocity (m/s)	Brownian displacement per second (m)
250	407	417.5	4.977	4.979	4.36×10^{-7}
100	162.8	167.0	0.796	0.797	6.89×10^{-7}
50	81.4	83.5	0.199	0.199	9.74×10^{-7}
40	65.1	66.8	0.127	0.128	1.09×10^{-7}
30	48.8	50.1	7.17×10^{-2}	7.19×10^{-2}	1.26×10^{-6}
20	32.6	33.4	3.18×10^{-2}	3.20×10^{-2}	1.54×10^{-6}
10	16.3	16.7	7.96×10^{-3}	8.04×10^{-3}	2.18×10^{-6}
5	8.14	8.35	1.99×10^{-3}	2.03×10^{-3}	3.08×10^{-6}
2	3.26	3.34	3.19×10^{-4}	3.33×10^{-4}	4.84×10^{-6}
1	1.63	1.67	7.96×10^{-5}	8.69×10^{-5}	6.89×10^{-6}
0.5	0.814	0.835	1.99×10^{-5}	2.35×10^{-5}	9.74×10^{-6}
0.1	0.163	0.167	7.96×10^{-7}	1.52×10^{-6}	2.18×10^{-5}
0.05	0.081	0.084	1.99×10^{-7}	5.63×10^{-7}	3.08×10^{-5}
0.02	0.033	0.033	3.19×10^{-6}	1.77×10^{-7}	4.87×10^{-5}
0.01	0.016	0.017	7.96×10^{-9}	8.08×10^{-8}	6.89×10^{-5}

2.2.1.2 Coagulation

Particles of dust in air will coagulate spontaneously and continuously, irrespective of the substance of which they are composed. As soon as they touch they will fuse and stick together. This process continues until the particles become large enough to settle.

The rate of coagulation increases with the turbulence of the medium in which they are contained. As eddies and swirls are formed the velocities of the particles relative to each other becomes greater, increasing the chance of collision and therefore increasing the rate of coagulation.

2.2.1.3 Impingement

Impingement (impaction) of dust particles occurs when an obstruction to the flow is encountered and the velocity of the medium is great enough. This impaction occurs when the inertia of the particle is high enough to cause the particle to stick to the obstruction. There is also the chance that high velocities will sweep the particles from the obstruction.

The impaction of particles on falling water droplets can be used for the removal of airborne particulate (dust) from the airflow. The success of this method is dependant upon the number, concentration and collision efficiency with the efficiency dependant upon size, density and velocity of the particle and the water droplet.

The size of dust particles is measured and expressed in microns (μm). The smallest dust particle that can be seen with the naked eye under good conditions (i.e. black on white with good lighting) is around $25\mu\text{m}$. It is generally accepted that the smallest particle visible to the naked eye can be as large as $50\mu\text{m}$.

Dust of particular concern to mining practitioners is that fraction inhaled and retained in the respiratory system. The site of the deposition varies with the size, shape and density of the

particles and this was first described in the proceedings from the Pneumoconiosis Conference held in Johannesburg in 1959.² Deposition in at these sites was shown to depend on three factors:

- (1) the percentage removal of particles before reaching the pulmonary lobules as the inhaled air passed through the nasal chamber and air passages;
- (2) the fraction of tidal air volume which reaches the pulmonary spaces;
- (3) the dust collection efficiencies of the pulmonary spaces.

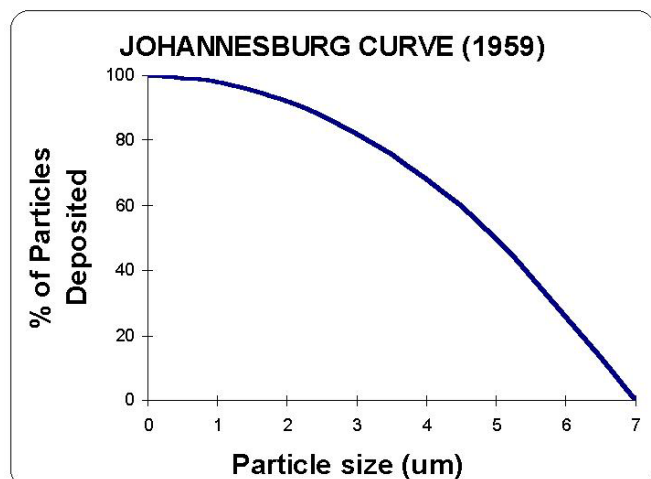
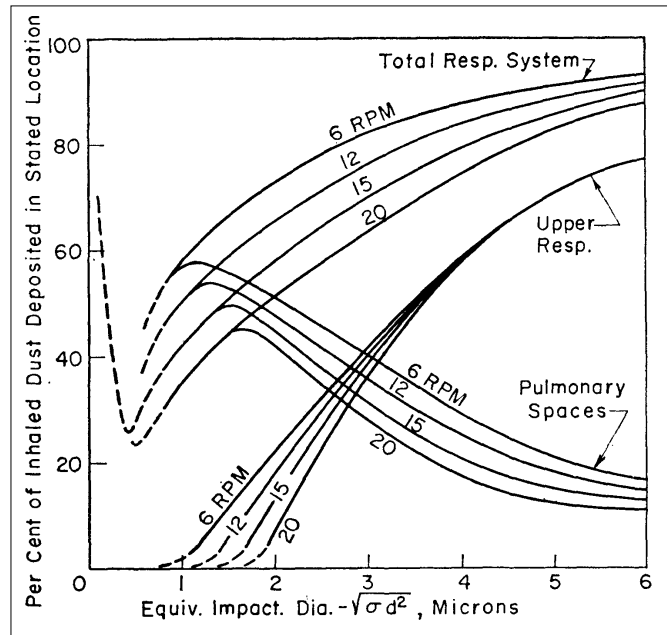
In short this experimental work shows that:

- (1) Particles with a diameter larger than $10\mu\text{m}$ are deposited in the nasal passages. The proportion deposited falling off as the size of the particle decreases. Virtually no $1\mu\text{m}$ particles are deposited in the nasal passages.
- (2) Particles with a diameter larger than $2\mu\text{m}$ tend to be deposited in the branching ducts leading to the lungs (bronchial tubes). For particles below $2\mu\text{m}$ there is insufficient time for settlement to occur. Below $0.5\mu\text{m}$, the probability for deposition increases due to the bombardment of these very fine particles by molecules.
- (3) Because of the high retention time in the lungs, the remaining particles have a high probability of deposition.

Recommendations from the 1959 Johannesburg Pneumoconiosis Conference have formed the basis for dust measurements monitoring in the years following. One of the most significant recommendations adopted from this conference was:

"That measurements of dust in pneumoconiosis studies should relate to the 'respirable fraction' of the dust cloud, this fraction being defined by a sampling efficiency curve which depends on the falling velocity of the particles and which passes through the following points: effectively 100% efficiency at 1 micron and below, 50 % efficiency at 5 microns, and zero efficiency for particles of 7 microns and upwards; all sizes refer to the equivalent diameters."

*(the 'equivalent diameter' of a particle is the diameter of a spherical particle of unit density having the same falling velocity in air as the particle in question.)"*³



² HATCH, T., "Respiratory Dust Retention and Elimination" Proceedings of the Pneumoconiosis Conference. Johannesburg 9th – 24th February 1959. p 113,132)

A simplification of the dust deposition curve was constructed on this basis and recommended by the International Pneumoconiosis Conference held in Johannesburg, South Africa in 1959. This curve has since become known as The Johannesburg Curve and clearly demonstrates that 100% of particles $<1\mu\text{m}$ and, 50% of $5\mu\text{m}$ particles and, 20% of $6\mu\text{m}$ particles, and, 0% of particles $> 7\mu\text{m}$ will enter the human lung, therefore it is the dust particles less than $7\mu\text{m}$ that are the primary concern for mines.

A number of health problems may be caused by the excessive and or prolonged inhalation of dust. The biological response is dependent upon the physical, chemical and toxicological properties of the dust involved.

Dusts that cause little or no chemical reaction and subsequent tissue scarring are classified as inert, although this may be misleading, as excessive inhalation will result in accumulation in the alveoli region causing mobilisation of the macrophages (which engulf small particles) and are the last line of defence of the respiratory system. Continued accumulation may eventuate with the formation of plaques on the alveoli walls causing shortness of breath and an increase in the frequency of colds and influenza etc.

Chemically active dusts (e.g. silica) decrease the active life of the macrophages and eventually result in permanent alteration or destruction of the alveoli. These permanent changes cause scarring of the lung tissue and result in a reduction of the elasticity of the lung tissue and a subsequent less efficient oxygen intake. This becomes very evident with a decreased lung expansion, breathlessness and a lessened capacity for work.

The long term effect of dust inhalation is not necessarily confined to the respiratory system and there is some research that suggests that the respiratory system merely provide the mode of entry into the body, other modes being absorption and ingestion. The respiratory system is very selective with respect to the size and quantity of dust retained in the lungs. It is not simply a matter of the quantity of dust in the atmosphere that dictates the amount deposited in the lungs, it is also the duration of exposure and the rate of deposition to the critical area.

2.2.2 Effects of Dust

Because the pre filters of the respiratory system are not developed to cater for the much finer dusts produced with mining activities, some of this dust manages to enter the finer passages of the lungs and remain there. It is this accumulation of dust in the lungs, which causes the condition known as pneumoconiosis (from two Greek words meaning dust and lungs).

The deposition in the respiratory tract occurs according to the following mechanisms:

- a) Gravitational settlement as determined by the free falling terminal velocity,
- b) Impacting due to inertia,
- c) Diffusion as applied to microscopic particles and,
- d) Interception due to the physical size of the particles.

Of those particles, less than $5\mu\text{m}$ only 50% penetrate into the alveoli region of the lungs. About 50% of these particles (usually less than $3\mu\text{m}$) deposit mainly due to diffusion and impaction. The remaining 50% will remain airborne and will be exhaled.

The terminal settling velocity of airborne fibres is almost independent of length. Deposition from gravitational settlement and impaction may be avoided in the upper respiratory tract allowing the fibre to penetrate into the fine air passages of the lungs. In these areas the length becomes the controlling factor for deposition. Interception becomes the mechanism. Fibres of $50\mu\text{m}$ and up

³ Recommendations Adopted by Pneumoconiosis Conference - Proceedings of the Pneumoconiosis Conference. Johannesburg 9th – 24th February 1959. p 619

to 200µm have been found in human lungs, the diameter of these fibres is almost always less than 3µm. The longest fibres are usually found in the bronchi and alveoli. In short, the probability of a fibre impacting the wall of an airway becomes greater as the fibre becomes longer.

Fibrous materials include asbestos, calcite, mica, magnesite, apatite gypsum and talc.

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2.2.3 Dust Exposure Standards

Allowable concentrations of dusts in work places have been determined from epidemiological studies that by their very nature extend over 20 to 30 year periods, or by experimental work on animals. This is because any physiological damage to workers is not usually detected in the early years of exposure and respiratory problems manifest only after a number of years of exposure. As a consequence exposure standards are constantly being reviewed and updated.

A number of health problems may be caused by the excessive and/ or prolonged inhalation of respirable dust. The biological response is dependent upon the physical, chemical and toxicological properties of the dust involved. Diseases caused by exposure to various types of dust can range from lung tissue scarring to pneumoconiosis, mesothelioma and lung cancer.

Reference should be made to <http://www.nohsc.gov.au/databases/> for the most up to date information on exposure standards. Some current dust exposure standards from this internet site which are relevant to the mining industry are:

Selected Exposure Standards (FROM [HTTP://WWW.NOHSC.GOV.AU/DATABASES/](http://www.nohsc.gov.au/databases/))

Dust Type	Exposure Standard (TWA)
Quartz	0.2 mg/m ³ (respirable)
Coal Dust	3 mg/m ³ (respirable)
Asbestos including: Amosite, Crocidolite, other forms or mixtures.	0.1 f/ml (respirable)
Chrysotile	1 f/ml (respirable)
Where there is no exposure standard	10 mg/m ³ (inspirable*)

*refer to <http://www.nohsc.gov.au> for definition of "inspirable"

2.2.4 Explosive Dusts

The most common explosive dust in metalliferous mining is sulphide dust. Sulphide dust explosions can cause considerable damage to underground facilities and will cause the release of poisonous gasses (primarily SO₂).

Sulphide dust explosions generally occur at firing time in massive pyritic or pyrrhotitic ore bodies and have been documented since at least the early 1920's. The explosions are known to occur in ore and waste rock containing 20% sulphur with reporting's of explosions in rocks with as little as 11%. These explosions are significantly more destructive when they occur in stope blasting than in development headings.

Because of the large quantities, high concentrations and toxicity of the sulphur dioxide gas produced with these explosions, re-entry into the mine is extended beyond what could normally be expected. Because of this the recommended approach is to eliminate the explosions by adoption of preventative measures prior to and during blasting.

The following procedures should be adopted in orebodies that are known to have the potential to produce sulphide dust explosions:

- Adequate stemming should be used
- The use of detonating cord should be eliminated
- All personnel should be removed to a safe place before firing

Enright⁴ & ⁵. (1996) when discussing preventative measures states "*Murphys Law*" applies with a vengeance in underground mining and it is inevitable that despite all reasonable precautions sulphide dust explosions will occur."

2.3 Heat

The adverse effects of heat range from discomfort, though to life threatening illness, such as heat stroke. Even relatively low levels of heat can lower the workforce morale, with all the attendant problems of high accident rates and low productivity. This simple cause and effect is often not fully appreciated and a number of mines attempt to "soldier on" without ever facing up to what can be a very important and significant problem which impacts on employee health, safety and mine productivity.

The principle for the control of heat build up in a mine, is the same as the principle of a car radiator. For example if a car is allowed to idle for extended period of time (such as when caught in a traffic jam in the middle of summer), the car engine will quickly overheat because there is insufficient airflow passing through the radiator. Air at normal atmospheric temperature flowing through the radiator will absorb some heat from it. If the air is constantly flowing through the radiator then the heat will be removed, and the radiator will be kept at a lower temperature. Up to a point the faster the flow the faster the heat removal.

The effect is similar in humans and machinery working in a confined space such as an underground mine. The surrounding air absorbs the heat generated by humans, machinery and the surrounding rock to a temperature equal to the temperature of the person, rock or machinery. Which ever is the highest?

Sources of heat in mines can be categorised as either natural (e.g. rock temperatures, ambient air temperatures and auto compression) or artificial (e.g. diesel and electrical powered equipment) and need to be considered at the early stages of any project.

As air passes through the mine ventilation circuit it is subject to a number of sources of heat. These include:

⁴ ENRIGHT, R.J., LEONTE, M. "AMIRA – Sulphide Dust Explosions Volume 1 – Cause and Prevention in Development Headings Project P316-P316A (1990-1994)" (11 May 1995)

⁵ ENRIGHT, R.J., LEONTE, M. "AMIRA – Sulphide Dust Explosions Volume 2 – Detection and Preventative Measures. Project P316A-P316B (1990 –1995)" (1996).

Auto Compression: - As air travels down the intake airways from the surface, its elevation decreases. There is a corresponding conversion of potential energy into enthalpy⁶. The magnitude of the change in enthalpy can be estimated using the steady flow energy equation for a flow from a higher elevation (Z_1) to a lower one (Z_2), assuming no heat flow and no work done:

$$H_2 - H_1 = g(Z_1 - Z_2) \quad \text{Equation 2-3 Auto Compression}$$

Where:

H = enthalpy (J/kg)

Z = elevation (m)

g = acceleration due to gravity (9.81 m/s^2)

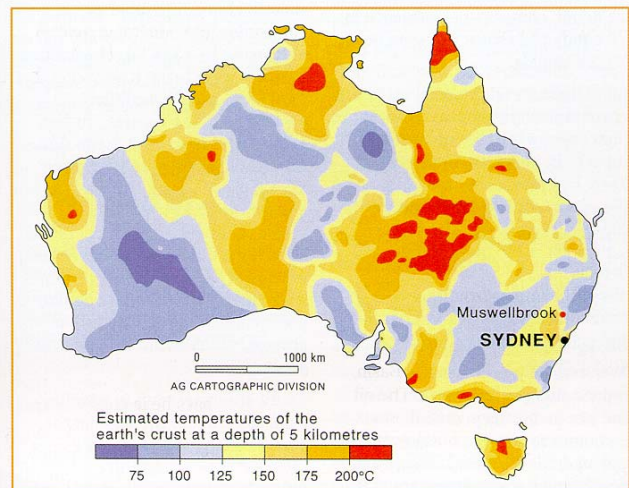
The enthalpy thus increases by 9.81 kJ/kg for every $1,000\text{m}$ decrease in elevation. For dry air, the thermal capacity is $1.005 \text{ kJ/}^\circ\text{C}$ and the theoretical dry bulb temperature increase is $9.81 \text{ kJ/kg/1,000m} \div 1.005 \text{ kJ/kg}^\circ\text{C} = \mathbf{9.76 \text{ }^\circ\text{C/1,000m}}$. In other words, the temperature of dry air flowing down a dry $1,000\text{m}$ shaft into a mine would increase by $9.76 \text{ }^\circ\text{C}$ (assuming there is no heat exchange between the air and the rock surrounding the shaft). The following points should be noted:

- Autocompression is not strictly speaking a heat source (it results from a conversion of energy, rather than from the addition of an external heat source).
- Autocompression causes the air temperature to increase, therefore as the mining depth increases, the ventilation air has less ability to remove heat.
- The temperature rise due to autocompression is independent of the airflow rate. In contrast, as the airflow increases, the temperature rise due to other sources of heat decreases.

It is also important to note that water temperature will increase with depth. If the water is contained in pipes then this increase in temperature is in the order of 0.2°C per 1000m . If the water is free flowing then this temperature increases to 2.34°C per 1000m .

Transfer from the surrounding rock:

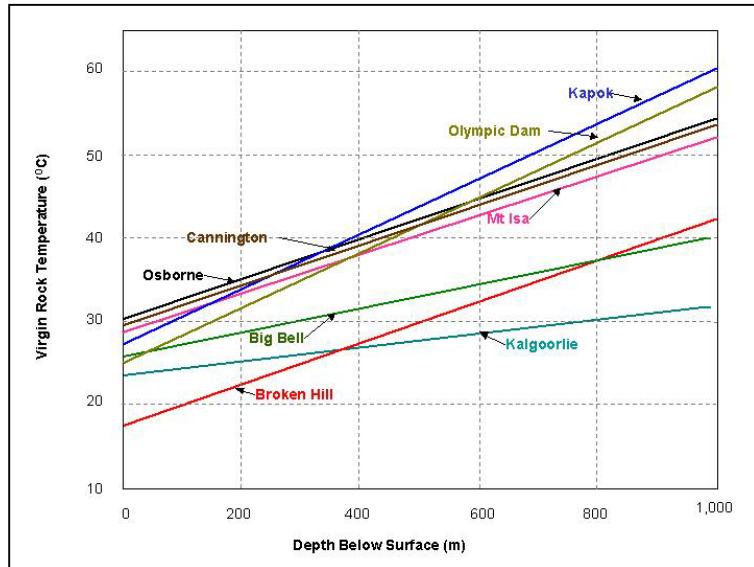
Surface rock temperature is around the annual average air temperature and can provide either cooling or heating depending on the air temperature passing over the rock. As we get deeper it gets hotter because the rock hasn't transferred its heat to the air. It does not always follow that cold places have cold rocks. In some places the rocks are hotter than others simply because they are newer and have yet to cool down and this varies considerably. For example the rock temperatures in Tasmania are hotter than the rock temperatures in the Barkley Tableland of Northern Territory.



Geothermal mapping reveals the hot spots deep beneath the Earth's surface.
Map: Courtesy Australian Geographic Cartography Division.

⁶ Enthalpy is the energy per unit mass (in this case of the air), resulting from the random motion of the air molecules. Elthalpy includes the thermal energy and the energy due to the pressure of the air.

The rate of temperature increase with depth is known as the geothermal gradient. The geothermal gradient varies depending on many factors, however typical gradients in Australia range from 1 to 3 °C per 100m of vertical depth. The surface rock temperature (i.e. 20 – 30m below surface, where temperatures are largely unaffected by surface climatic variations) is close to the average annual ambient dry-bulb temperature. For dry airways, the heat flow from the surrounding rock to the ventilation air is proportional to the difference between the virgin rock temperature and the air temperature. The rate of heat flow from the rock to the air increases when the airway is wet. The rock surrounding an airway has the ability to absorb and subsequently release thermal energy, depending on the difference between the rock temperature and the air temperature. This is sometimes known as the thermal flywheel effect. In some mines, oxidation of exposed minerals can also be a significant source of heat.



and subsequently release thermal energy, depending on the difference between the rock temperature and the air temperature. This is sometimes known as the thermal flywheel effect. In some mines, oxidation of exposed minerals can also be a significant source of heat.

Ground Water Ground water flowing from the rock into an airway acts to transfer heat from the rock to the air. The ground water temperature is almost always the same temperature as the virgin rock temperature. The amount of heat transfer can be limited by use of efficient pumping and drainage

practices (e.g. water should be collected in drains and kept off declines).

Machine Heat Except in cases of hoisting, hauling or pumping (where potential energy is raised), almost all of the output power of underground machines is used to overcome friction (i.e. it is converted into heat). Diesel engines are thermally inefficient and generate significant heat loads. At full power, they are about 33% efficient (i.e. 33% of the fuel energy value is converted to flywheel power (almost all of which eventually converts to heat anyway), the other 66% is converted directly to heat). For example, a diesel truck operating on a level gradient and producing 200 kW of engine power would emit about 600 kW of heat. In underground mines, secondary ventilation fans are also a major source of heat. For example, a fan consuming 180kW of electrical energy does no useful work in a thermodynamic sense and hence all of the 180 kW of electrical energy is converted to heat.

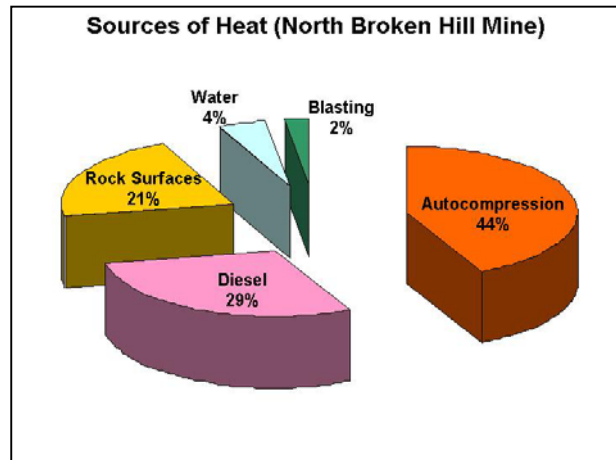
Explosives Only 5% of the energy produced by blasting is used to break the rock, the remaining 95% is released as heat. For many years it was thought that this heat was dissipated directly to the ventilation system and removed during the re-entry period. It is now more widely accepted that this heat is transferred to the broken rock and liberated over a much longer period of time and is variable, depending upon the ventilation rate and the rock surface exposure.

Cement Fill Only Hydraulic Fill would add significant quantities of water and therefore heat, to the mine environment. All of the other fill types are non-draining and, except for the flushing water used with Paste Fill, should have minimal impact. As cemented backfill cures in the stopes, heat is produced due to the exothermic reaction of the cement and water. Most of this heat will be absorbed into the surrounding rock and slowly released and generally this would be insignificant on a mine wide basis. However, it may be significant locally during the first few days after placement. If the paste is placed at a temperature of 30°C and rises by 10°C then this will raise the temperature of the surrounding rock to 40°C. In hotter mines heat may flow from the rock to the fill until equilibrium is reached. The radiant heat from the fill in this case would

almost be imperceptible. In the worst case some of the 'excess' heat would be released to the ventilating airflow and rejected to the exhaust ventilating system.

An appreciation of the proportion of the total heat load contributed by the various heat sources can be gained from the example opposite, which is based on heat load calculations performed for the North Broken Hill mine. At the time the calculations were performed, the mine was operating at a depth of about 1,600m below the surface.

It should also be noted that surface ambient conditions can also be an important contributor to underground heat conditions. This is particularly the case in northern Australia, where high surface temperatures and humidity can give rise to heat problems, even in relatively shallow mines.



2.3.1 Air Temperature

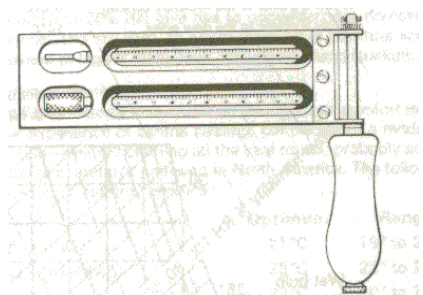
Temperature should not be confused with heat. Heat is a form of energy and is calculated. Temperature is a state and is measured. For example a bath full of water at 30° contains more heat than a cup of water at 70°. The difference being the thermal capacity which is the ability to raise the temperature of 1kg by 1°C.

There are two measurements of air temperature are important to ventilation practitioners, the dry-bulb and the wet-bulb.

Dry- bulb temperature is the actual air temperature, measured with a standard thermometer and the wet-bulb temperature is the measure of the evaporative capacity of the air. Together they measure relative humidity. Humidity is the ratio, expressed as a percentage, of the water vapour present in the atmosphere to the amount required to saturate the air at the same temperature. As humidity increases, the cooling from evaporation of sweat decreases.

The ability to measure both the wet and dry bulb air temperature is essential, particularly in hot or poorly ventilated mines, to monitor potential heat stress conditions. These two properties are also required to accurately determine the air density.

In mining, the most widely used instrument to determine wet and dry bulb air temperature is the whirling hygrometer (sling psychrometer), or wet and dry bulb thermometer. It consists of two identical thermometers (usually the mercury type) mounted side by side in a rigid plastic frame. The frame is attached to a handle via a spindle and is free to rotate. The thermometers are graduated from -5°C to +50 °C. The bulb of one thermometer is surrounded by a cotton wick that is kept moist by a small supply of distilled water in a reservoir. This is known as the wet-bulb.



The dry bulb thermometer provides the sensible temperature of the air. The wet bulb thermometer provides a measure of the evaporative rate of the air. When the air is dry, the moisture in the wet cloth will evaporate faster and the temperature will cool. When the air is humid, very little moisture evaporates from the wet cloth, and the cooling process slows. The smaller the difference between the two temperatures, the higher the humidity. If required, the

relative humidity can be determined by equating these two temperatures using a table or slide rule supplied with the instrument.

Equivalent electronic instruments are available, however, they do not compete with the simplicity and reliability of the sling psychrometer. In particular, the humidity sensors on electronic instruments seem to cause trouble unless very good quality, very expensive units are used.

2.3.2 Body Heat Balance

Food (fuel) is oxidised in the metabolic process and converted to energy, in the forms of:

- Metabolic heat
- Mechanical work, and
- Change in mass (body growth)

The latter is negligible (usually!) and can be ignored. Although metabolic heat production depends primarily on muscular activity (i.e. it is related to the rate of work), it also varies with:

- Condition of the individual's health,
- Physical fitness, and
- Emotional state

In essence, the human body is a biological engine of with low mechanical efficiency. Less than 20% (usually much less than 20%) of the available energy is converted into usable mechanical energy. That part of the metabolic process not used to provide the mechanical work (i.e. >80%) will always appear in the body as heat. This metabolic heat must be rejected from the body to the environment; otherwise the body's core temperatures will increase, possibly to life threatening levels. In fact, if none of the metabolic heat could be rejected, the body temperature would rise by 1°C (the maximum ISO recommended acceptable rise) in 12 minutes during moderate exercise and 4 minutes during strenuous exercise.

As we work we release energy in the form of heat and understandably the harder we work the higher is the metabolic heat generation process. The removal of heat generated by the human body is reliant upon our ability to sweat and the rate at which it can be evaporated.

This heat is transferred to the external environment and, if the rate of generation is greater than the rate of transfer the body temperature will rise. This heat storage is called the "metabolic heat generation" or the "metabolic heat production rate".

Heat transfer between the body and the environment occurs via:

- Respiration (breathing)
- Radiation
- Conduction
- Convection, and
- Evaporation

Respiratory Heat Exchange. In a typical hot underground environment, respiratory heat exchange accounts for about 5% of metabolic energy production. This heat loss is generally considered to be small enough to be ignored.

Radiant Heat Exchange

The magnitude and direction of radiant heat transfer depends on the temperature difference between the human body (skin) and the object. This type of heat transfer is usually only significant in mines with hot rock temperatures, and in the vicinity of hot diesel equipment.

Conductive Heat Exchange Conductive heat transfer occurs when two bodies come into contact. In normal mining activities, conductive heat transfer to or from the human body is usually negligible (unless, for example personnel are wearing cooling jackets).

Convective Heat Exchange Convection occurs when a layer of cool air comes in contact with warm skin. The air increases in temperature and its density decreases. As the air becomes lighter, it rises, taking the heat away from the skin. As the air is replaced with cooler, more dense air, the process of heat exchange continues.

Convective heat transfer to the surrounding air is typically 15% to 20% of total cooling. It depends on the temperature difference between the skin and the dry bulb temperature of the air, as well as the relative air velocity and body surface area.

Evaporative Heat Exchange Evaporation is the main cooling mechanism for the body, contributing about two thirds of total cooling. Evaporative cooling relies on the latent heat of vaporisation of sweat from the body. The effectiveness of this form of heat transfer is dependent upon:

- efficiency and rate of sweating and,
- the evaporative capacity of the environment (depends mainly on air temperature, humidity, air velocity, and type of clothing).

The ability of the body to sweat depends on physical fitness, acclimatisation (training the body to sweat efficiently) and on the body having sufficient fluid.

There is considerable variation in the sweat rate of individuals and this is partly associated with the degree of acclimatisation (it also depends on general health and genetic factors). Sweat that drips from the body, serves no cooling function and it begins to drip from the skin surface well before the body is fully wetted. This is primarily because some areas of the human body produce more sweat than others. Sweat begins to drip when the skin surface is approximately 50% wet.

In terms of maximising the cooling effect, the most effective place for sweat to evaporate is from the skin. Clothing (particularly clothing made of artificial fibres) can significantly reduce the amount of heat rejected from the body via evaporative heat exchange (it also reduces convective heat exchange).

In underground mines, the evaporative capacity of the environment can be increased primarily by increasing the air velocity. In some instances, it becomes necessary to also lower the wet bulb air temperature (e.g. by reducing moisture pick-up in intake airways or through the use of refrigerative mine air cooling).

2.3.3 Thermoregulation

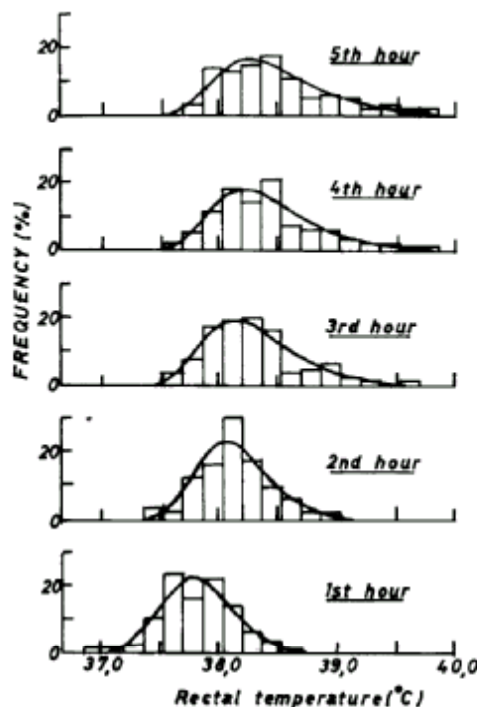
The core of the body must be maintained at a stable temperature of between about 35°C and 40°C. To do this, the body invokes its thermoregulation processes. These include:

- **Cold** – Skin blood flow is reduced to reduce heat losses to environment. Shivering commences to raise the metabolic rate.
- **Heat** –Skin blood flow is increased to increase heat transfer rate. Sweating increases to maximise evaporative heat loss.

The effectiveness of the body's thermoregulation system is dependent on a large number of factors, including:

- **Fitness**
- **Age**
- **Obesity** results in excessive fat insulating the body, reducing heat loss
- **Acclimatisation** is the adaptation of the body's thermoregulatory systems to working in hot conditions. Most of the effects of acclimatisation are generally developed within a week of working in hot conditions, but the process continues for at least 14 days. Conversely, acclimatisation is lost after a period of 7 to 14 days away from hot conditions. The degree of acclimatisation is related to the level of heat stress experienced on the job. Some personnel are heat intolerant and can never be successfully acclimatised to hot working conditions.
- **Hydration Levels** are critical. Even a small decrease in hydration levels will lead to a substantial reduction in the ability to work effectively in heat. Significant fluid intake is required to counter losses through sweating in hot conditions. Hydration levels can also be compromised by diuretics (alcohol, caffeine etc) and also by illness.
- **Clothing** acts as insulation, reducing the body's ability to reject heat to the environment.

The illustration below shows the Temperature Response to Heat Stress there is a tremendous spread in the effectiveness of the body's thermoregulation system, even across a relatively narrow sample group (in this case, 99 acclimatised, essentially nude men working at the same rate in the same environment).



Hourly frequency distributions of rectal temperature for 99 acclimatized men exposed to a single combination of work rate and environment. The curves follow a 3-parameter log-normal distribution law.

Obtained from Chapter 20 of "Environmental Engineering in South African mines", published by the Mine Ventilation Society of South Africa in 1989.

2.3.4 Heat Related Illness

It is important to be able to recognise the causes and symptoms of overexposure to heat and to know the treatment for these illnesses. Over-exposure to heat occurs when the body's temperature controlling mechanism (thermoregulation) cannot cope with the thermal environment and rate of exertion. Over-exposure to heat may quickly progress to collapse, unconsciousness (coma) and death.

The symptoms and treatments for some of the many classifications of heat illnesses are listed below (in order of increasing severity):

Heat Cramps result from an imbalance in the body electrolytes, caused by vigorous activity, dehydration and high temperatures. The body loses more fluids than it is replacing. This fluid deficit causes muscles to lose their vital electrolyte balance (complex salts), thus causing muscular contraction (cramps).

Signs and Symptoms	Treatment
Pale, clammy skin. Sweating Cramping pains (in the limbs and/or abdomen). Nausea. Spasms (in the affected limb or limbs).	Rest the victim in a cool location. Give sips of water to drink (after nausea has passed). Don't massage affected limbs. Discourage any further exercise.

Heat Exhaustion occurs after prolonged moderate elevations of core temperature. Its development is usually attributed to the inability of the circulation system to meet the demands of thermoregulation (i.e. the diversion of significant quantities of blood flow to the skin) whilst also maintaining sufficient blood flow to the vital organs (brain and skeletal muscle).

Signs and Symptoms	Treatment
Pale, clammy skin. Restlessness Cramps in the limbs and/or abdomen Nausea and/or vomiting Headache Weakness Fatigue	Rest the victim in a cool location Discourage any further exertion. Cool down casualty by sponging. (use tepid water) Give cool water to drink (cautiously, after nausea has passed).

Heat Stroke is a very serious condition known as a Core Temperature Emergency. It occurs as a result of thermoregulatory failure. If appropriate treatment is not instigated promptly, heat stroke carries a mortality rate of up to 80%. High body temperatures associated with heat stroke can result in irreversible damage to organs (especially the brain, kidneys and liver) and the nervous system.

The main causes of heat stroke in the mining industry⁷ are overly strenuous work in hot environments and dehydration (often associated with excessive alcohol consumption).

⁷ Mining Industry heat stroke in countries such as South Africa. **Note** that it is unlikely that there has ever been an underground mining fatality directly attributed to heat stroke in Australia.

Signs and Symptoms	Treatment
Body core temperature above 40.5°C Often a cessation of sweating. Aggressive or irrational behaviour Staggering, Dizziness or Faintness Vomiting Collapse and seizures Coma	Cool down the casualty <u>immediately</u> by gently splashing with cool (but not icy) water. If possible, increase evaporative cooling by fanning patient. Continue treatment until medical help arrives. Give frequent small drinks of water if patient is conscious. Prepare to resuscitate if required.

2.3.5 Heat Stress Indices

Measurement of hot conditions can either be by measuring the climate, that is the cooling power of the environment or by measuring the heat strain, the effect on persons working there.

The four main parameters for directly evaluating heat strain are body core temperature, heart rate, skin temperature and, weight loss through sweating. Although all of these can be measured it is not practical to do so in the workplace. Even if heat strain could be measured accurately, it does not indicate a reason for the problem and hence any possible solution.

The “degree” of heat stress is a function of the parameters outlined in the table below.

Parameters Contributing to Heat Stress

Metabolic rate
Ambient and radiant temperature
Water vapour pressure
Air velocity
Barometric pressure
Amount and type of clothing
Skin surface area and “view factor” (e.g. whether sitting, standing etc)
Variability in human thermoregulatory response.

Development of a measure to determine safe levels for work in hot places has been debated over many years. During the 1900’s over 90 heat stress indices have been developed and used around the world with varying success and acceptance by the international community. Generally these indices combine one or two parameters into a single number and therefore only partially represent the complexity of the human thermoregulatory system and the climatic conditions encountered.

Broadly heat stress indices can be classified into three types: -

- Single measurements
- Empirical methods, and
- Rational indices

2.3.5.1 Single Measurements

Although there has been some attempt to use a single measurement to determine heat stress conditions, there is no single parameter that provides a reliable indicator of physiological reaction.

2.3.5.1.1 Psychometric Wet-bulb Temperature

For many years, the wet-bulb temperature has generally been considered as the easiest and simplest measure of heat stress and has been used widely in many Australian mines as the sole indicator for climatic acceptability.

Australian mining legislation uses a wet-bulb temperature of 25°C, 27°C or 28°C as a trigger point for modified work conditions such as reduced work hours or the introduction of regular rest periods. 32°C is usually set as the upper limit for work being allowed to continue.

The psychometric wet-bulb temperature is a measure of the evaporative cooling power of the environment and is therefore of limited value especially in high air velocities and high radiant temperatures.

It worthy to note, that in wet-bulb temperatures above 37°C, the environment is unable to support human life for any extended period of time.

2.3.5.1.2 Dry-bulb Temperature

Dry-bulb temperature above 45°C can give a burning sensation to exposed skin and is generally accepted as the upper limit for work being allowed to continue.

2.3.5.2 Empirical Methods

The most commonly used empirical methods have been, effective temperature (ET), Kata thermometer, predicted four-hourly sweat rate, and wet-bulb globe temperatures index (WBGT).

2.3.5.2.1 Effective Temperature (ET)

ET was developed in 1923 as a measure of comfort by the American Society of Heating and Ventilating Engineers and primarily for use in offices. The ET of an environment is the temperature of a saturated environment without movement of air that would produce the same instantaneous thermal sensation as the environment being considered. As this index is based on subjective thermal sensation it has shortcomings in either low (<0.5m/s) or high (>3.5 m/s) air velocities.

Although the value can be calculated, it is normal to refer to empirically constructed nomograms such as :-

1. Basic scale developed for essentially nude men and,
2. Normal scale for lightly dressed men.

It is not possible to reduce these nomograms to a simple mathematical expression over the entire range.

The Basic ET may be calculated to within 0.2°C from the following equation given:-

- The difference between the wet-bulb and dry-bulb temperature is less than 5°C
- The wet bulb temperature is within the range from 25°C to 35°C, and
- The velocity of air is within the range from 0.5 m/s to 3.5 m/s

$$ET = 20.86 + 0.0354T_{WB} - 0.133V + 0.07V^2 + (4.12 - X_1 + X_2) / 0.4129$$

Where

$$X_1 = [8.33 \{ 17X_3 - (X_3 - 1.35)(T_{WB} - 20) \}] / [(X_3 - 1.35)(T_A - T_{WB}) + 141.6]$$

$$X_2 = 4.25 [(T_A - T_{WB})X_3 + 8.33(T_{WB} - 20)] / [(X_3 - 1.35)(T_A - T_{WB}) + 141.6]$$

$$X_3 = 5.27 + 1.3V - 1.15e^{-2V}$$

T_A = Ambient dry-bulb temperature ($^{\circ}\text{C}$)

T_{WB} = Ambient wet-bulb temperature ($^{\circ}\text{C}$)

V = Velocity of air (m/s)

e = vapour pressure of the air (kPa)

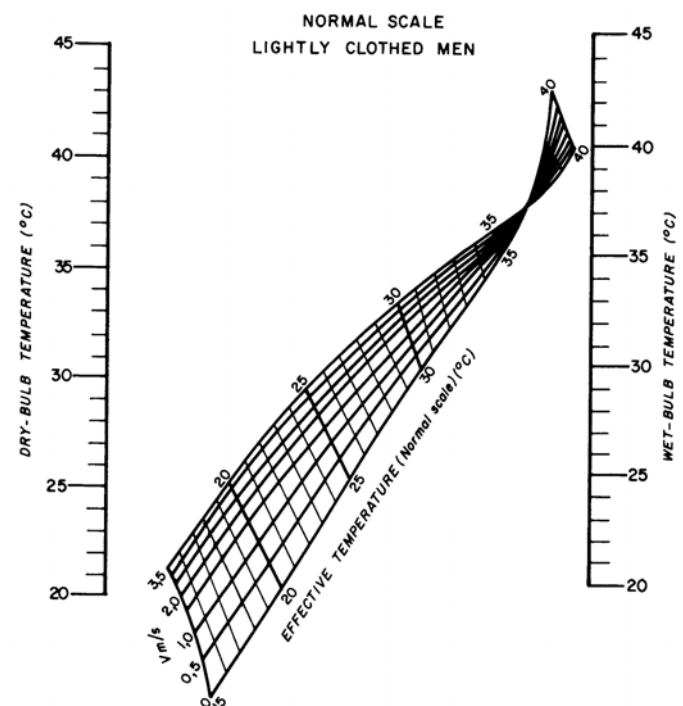
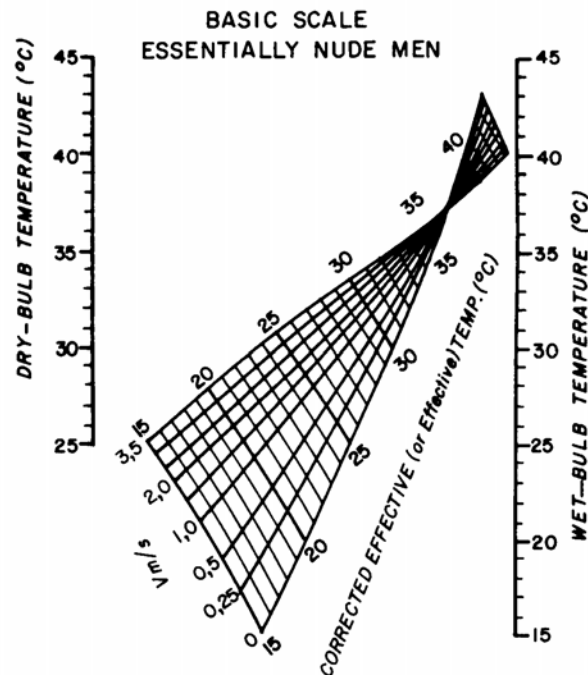
Subsequent to the development of these nomograms, it has become customary to use the temperature of a blackened 150 mm hollow copper sphere in place of the dry bulb temperature. The assumption is to take into account the effects of thermal radiation. Measurements using the blackened sphere are termed corrected effective temperature.

ET has been extensively used in the European and British mining industries. In Queensland the Coal Mining Act – General Rules for Underground Coal Mines Part 2.7 (1) (a) provides that no person shall be employed where the effective temperature in the workplace is or exceeds 29.4°C and Part 2.7 (3) specifies a shortened shift strategy when the ET exceeds 27.2°C .

The use of ET as a tool for prevention of heat related illness has a number of problems

1. ET exaggerates the effects of thermal sensations at high relative humidity
2. Only accounts to climatic conditions and does not consider work rates
3. ET scales are most accurate in warm climates and low heats stress conditions.
4. ET is least accurate in velocities less than 0.5 m/s and, greater than 3.5 m/s.

2.3.5.2.2 Kata Thermometer



2.3.6 Air Cooling Power

A complete and supportable heat stress index must account for all of the contributing parameters. Currently, the most complete and supportable index of heat stress is the concept of air-cooling power, which was developed by the South African Chamber of Mines. This index was however specific to South African mining practices (in particular, it assumed personnel were essentially nude and fully wetted by perspiration).

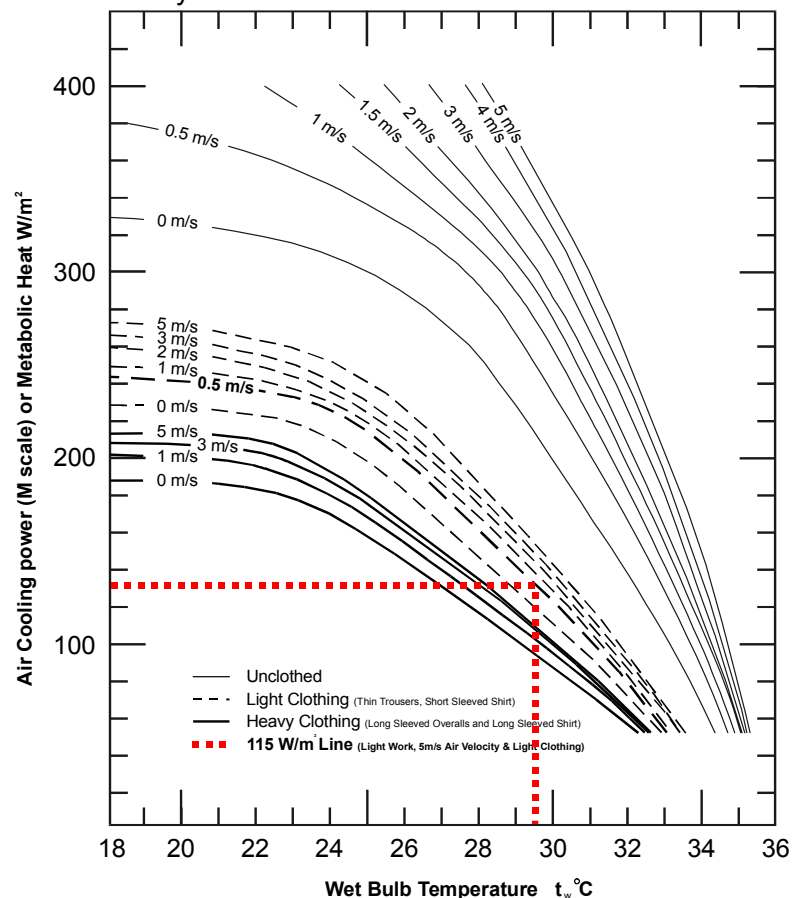
There have been a number of updates and modifications to the original scales and it is important to state *which* air cooling power scale is being used. McPherson (1992)⁸ describes how to calculate air-cooling power, referred to as the “M” (McPherson’s) Scale Air Cooling Power.

The concept of air-cooling power relies on the quantification of the ambient environment’s ability to remove metabolic heat from the human body. The scale used to determine the rate of generation of heat by the human body and also air-cooling power is W/m^2 of body (skin) surface area⁹.

By definition, provided the air cooling power is equal to, or greater than the metabolic rate, then there will be less than a one in one million chance of the healthy, acclimatised and “self pacing” individual developing dangerous body core temperatures ($>40^{\circ}C$), potentially leading to heat stroke. This is held to be an “acceptable” risk.

A widely used thermal acceptance criterion is a minimum air cooling power of $115 W/m^2$ (i.e. if the air cooling power is less than $115 W/m^2$, then an individual could not even sustain a “moderate” work rate without incurring an unacceptable (greater than one in one million) chance of a healthy acclimatised individual suffering from heat stroke.

Whilst air-cooling power is the most complete heat stress index, it requires fairly complex calculations (ideally requiring a computer) to solve. The calculations require inputs including; amount and type



Radiant Temperature = Dry Bulb Temperature
 Dry Bulb Temperature = Wet Bulb Temperature + $5^{\circ}C$
 (Note that the graph may be used without undue error
 for differences between wet bulb and dry bulb temperatures of between 2 and $8^{\circ}C$)

Figure From “The Generalisation of Air Cooling Power” by M.J. McPherson
 Fifth International Mine Ventilation Congress The Mine Ventilation Society of South Africa, Johannesburg, 1992.

⁸ “The Generalisation of Air Cooling Power” M.J. McPherson. Fifth International Mine Ventilation Congress The Mine Ventilation Society of South Africa, Johannesburg, 1992.

⁹ For reference, the average skin surface area of a 1.7m tall, 60.5 kg South African miner has been determined to be $1.8 m^2$. The modified DuBois formula, relating skin surface area (A_s , m^2) to body mass (m , kg) and height (h , m) is $A_s = 0.217m^{0.425}h^{0.725}$ (From “The Mine Ventilation Practitioner’s Data Book” Second Edition, Andrew Patterson et. al. 1999 The Mine Ventilation Society of South Africa, Johannesburg).

of clothing worn, wet bulb temperature, dry bulb temperature, radiant temperature of surroundings, air velocity, barometric pressure and the work rate of the individual. The large number of required inputs and requirement for a computer to calculate air-cooling power has limited its use to date as a practical tool for determining on the spot whether the thermal conditions in an underground environment are “acceptable”. Recently this problem has to a large extent been overcome with the development of a robust, portable heat stress meter designed for use underground. The meter uses algorithms based on air-cooling power and is based on an instrument that was originally developed for use by the US military in “Operation Desert Storm” in Kuwait.

For mines which do not have access to the heat stress meter, it should be noted that normal ranges of some of the factors mentioned above in the underground mining context have a relatively weak effect on air cooling power. A generalised Air Cooling Power Chart that assists in the rapid manual assessment of the acceptability of the thermal environment can be produced on this basis. (See McPherson’s “M” Scale Air Cooling Power Chart opposite) The following assumptions were made in order to produce such a chart:

Typical Metabolic Work-rate Classifications for Healthy Adults

Light Work < 115 W/m ²	Moderate Work 115 to 180 W/m ²	Hard Work 180 to 240 W/m ²	Very Hard Work >240 W/m ²
Sleeping 40 Seated 60 Standing 70	Walking 5 km/h Trades people Jumbo drilling Diesel Operator	Walking 6.5 km/h Building brick walls Scaling Hand-held drilling	Shovelling Timbering

Barometric pressure – Assume $P = 100$ kPa. Air cooling power is largely unaffected by normal range of pressures found within underground mines.

Radiant temperature of surroundings is equal to the dry-bulb temperature (This is usually the case, except near hot surfaces such as diesel radiators etc).

Dry-bulb temperature is equal to the wet-bulb temperature + 5°C (Note that the graph may be used without undue error for differences between wet bulb and dry bulb temperatures of between 2 and 8°C.) The majority of underground temperatures except near diesel radiators etc will fall within this range.

Clothing - Assumptions were also made about the thermal resistance and area factor of different clothing specifications and also regarding the body posture factor.

The above assumptions allow an air-cooling power graph to be produced, which only considers wet-bulb temperature, air velocity (air speed over the individual), and clothing type.

- The protective clothing worn underground in Australian operations would fall somewhere between the “Light” and “Heavy” categories.
- An example has been outlined with the heavy dashed line in the “M” scale chart. It assumes “light” clothing, “light” work rate (115 W/m²) and air velocity of 0.5 m/s. This line projects down to a wet-bulb temperature of 29.5°C. In other words, at the conditions and work rate outlined, a wet bulb temperature of less than 29.5°C will ensure that there is less than a one in one million chance of heat stroke occurring in a healthy, acclimatised individual.

In any mining operation, there will be variation in many of the variables listed earlier. It can however be concluded that based on all the assumptions implied in McPhersons ‘M’ Scale, an acceptable air cooling power can not be provided for even a “light” work rate at wet bulb temperatures above 32 °C.

It is also important to note that for a range of circumstances (e.g. heavier work rates), conditions which could lead to the development of heat stroke could occur at wet bulb temperatures **below** 32°C. For the purposes of providing a **simple, practical** measurement to determine the thermal “acceptability” of an underground environment, a **“stop work” cut-off of 32°C wet bulb is supported**. This temperature cut-off is combined with a requirement that the air velocity must be greater than 0.5 m/s at wet bulb temperatures over 25°C (e.g. refer to W.A. Regulations).

2.3.7 Management of Hot Working Conditions

Human heat stress is a health hazard that can be managed by:

Instigating “working in heat” protocols

Controlling the underground ambient heat conditions

Hot working condition protocols can include the following aspects:

Worker training and education – e.g. how to recognise and treat the symptoms of heat illness, the importance of drinking sufficient water, the need to “self-pace” according to conditions and the mechanism of acclimatisation. “Refresher” training courses should be re-run on a regular basis (ideally just before the onset of summer)

Hydration testing – e.g. Urine tests on personnel who work in “hot” areas of the mine. The tests quickly show those who are not coping with hot working conditions (e.g. those not drinking sufficient fluid, those who are ill or those who are heat intolerant). The protocols should spell out the policy regarding those who fail the hydration test (second chance etc).

Staff selection. Some mines preclude those with certain medical conditions, those who are overweight, personnel over a specified age and females with “child bearing capacity” from working in thermal conditions which are beyond a specified level.

“Stop work”. It is very important that a cut-off criterion with respect to acceptable/unacceptable underground ambient conditions is established. The criterion must be fully supported by management and strict guidelines established to ensure that the criterion is respected under all but exceptional circumstances (and these should be defined in the protocol).

Establishing working in heat protocols is of fundamental importance, however it only deals with part of the problem. In a number of deeper mines (or shallower mines hosted in hot rock), it is very difficult to maintain ambient conditions that allow even a light rate of work without incurring an unacceptable risk of the development of heat stroke. In these mines, it is necessary to boost the miners’ ability to reject metabolic heat by employing one or more of the following methods:

Increase the air velocity to improve the air-cooling power. This is effective at lower wet bulb temperatures. As the wet-bulb temperature increases, a “diminishing return” effect is apparent. In fact, again referring to McPhersons ‘M’ Scale, it can be seen that increasing the air velocity has very limited effect on the air cooling power, once the wet bulb temperature exceeds about 32°C. It should also be noted that there is a considerable cost penalty associated with increasing the airflow rate:

$$\text{Power Cost Increase} \propto \left(\frac{\text{New Flow Rate}}{\text{Old Flow Rate}} \right)^3$$

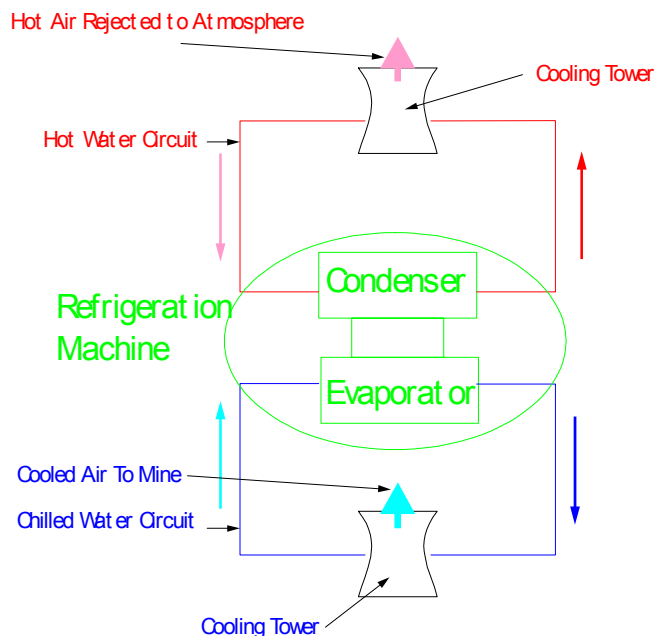
Microclimate Cooling. This involves cooling the localised area around the miner. One example is the use of air-conditioned cabins on mobile equipment. Another example is the use of cooling jackets (ice vests). These use flexible frozen gel-packs (similar to those used for sports injuries), which special pockets in an insulated vest. They are primarily intended for use in emergency



situations (and for sports competitors), but may potentially have some applications for more regular underground use. The gel packs are cooled in a chest freezer (in the crib room) and have a useful life of about 2 hours. The figure opposite is an illustration of a Micro-climate cooling vest. (Reference <http://www.steelevest.com/>). This is seldom a satisfactory “total” solution, since there is a very real chance of something going wrong (e.g. forgetting to change the cooling jacket gel-packs and subsequently getting “stuck” in a hot mining area).

Refrigerative Mine Air-cooling In spite of the sometimes considerable capital and operating costs, refrigerative cooling of mine air is often the only supportable option for control of heat stress in very hot mines. A simplified line diagram showing a “typical” mine refrigeration circuit is shown below.

The refrigeration machine generally consists of a screw compressor coupled to plate type heat exchangers (condenser and evaporator). The refrigerant is usually ammonia for surface plants and R134 for underground plants. Rated capacity of machines used in Australian Mines ranges from about 500 kW to 10 MW of refrigerative cooling capacity. As a rule of thumb, the compressor motor power draw is 1/4 to 1/5th the nominal refrigerative cooling capacity. The amount of (potable) water circulated in the plants can be considerable. For example, a 1MW plant circulates about 25 l/s in each of the (hot and cold) water circuits. A proportion of this flow evaporates and some is also dumped. Make-up water requirements for a 1 MW plant are typically 1.5 l/s in the hot water circuit and 1 l/s on the cold-water circuit.



Several plant configurations are possible:

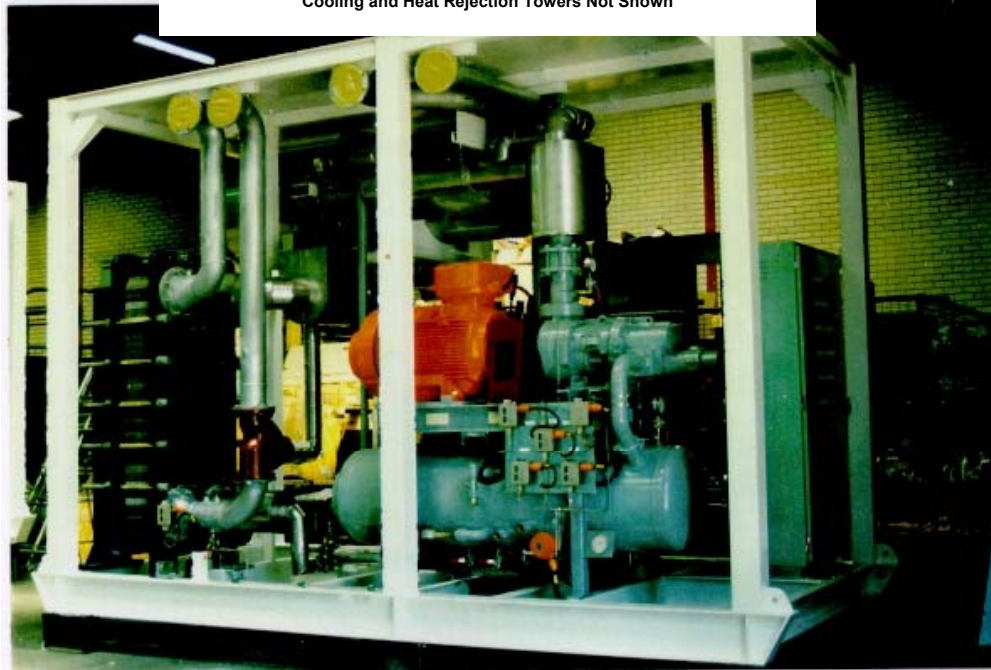
Surface Bulk Air Cooling. All of the equipment is located on the surface. The air is cooled at the intake raise collar. This system is simple and easy to maintain. It is not well suited to mines with complex ventilation systems (some of the cooled air may find its way to locations where chilled is not required). Some of the “coolth” is also inevitably lost in the mine intake airways.

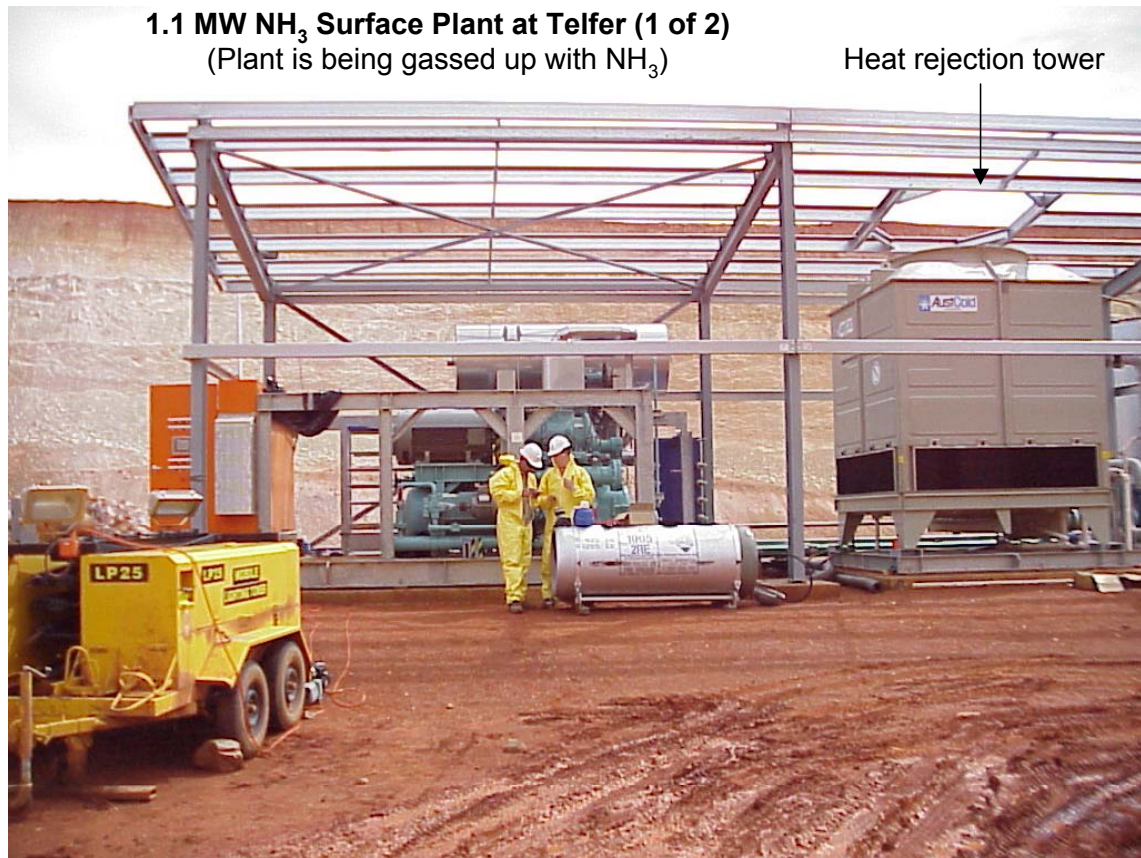
Surface Plant with Underground Coolers. With this system, the plant and heat rejection towers are mounted on the surface. Chilled water is reticulated underground to cooling towers that are close to the locations where cooling is required. Advantages of this system are that cooling can be more precisely and more directly delivered to the required locations. The disadvantages are high pumping costs (high volume, high head pumping although some energy losses can be recovered with the use of a Pelton wheel turbine coupled to a generator) and significant maintenance costs (complex system with cold and hot water storage dams insulated chilled water pipes, batch cooling of water, large underground pump stations and Pelton wheels, underground cooling towers which can be fouled with dust etc).

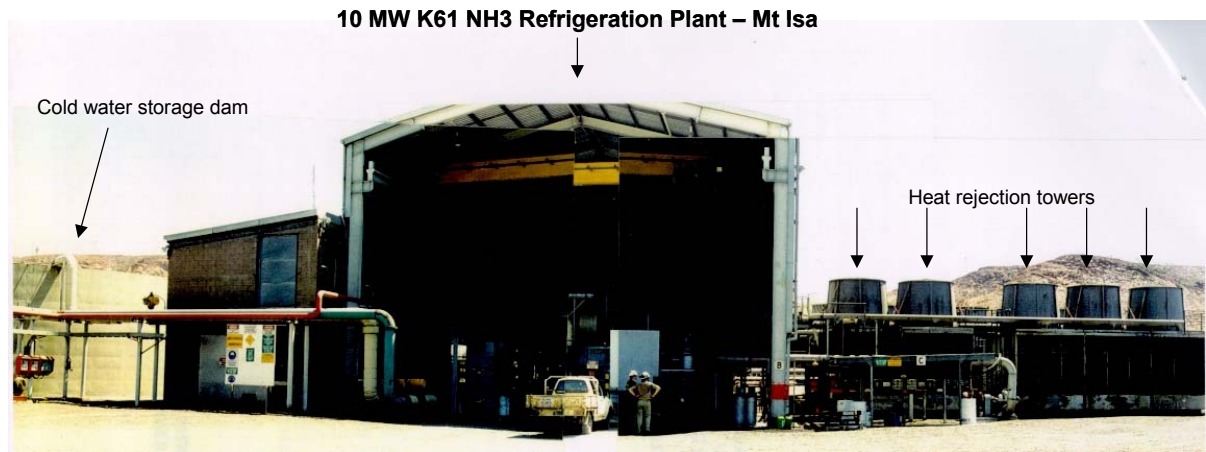
Underground Spot Cooling. All of the equipment is located underground. The refrigeration plant and heat rejection tower are located in a return airway. The cooling tower supplies chilled air to a specific location (often to the inlet of a secondary ventilation fan). The capacity of refrigeration machine is restricted to a size which can be physically transported underground on a skid (e.g. maximum of about 1MW cooling capacity). As a result, the machine generally only has sufficient capacity to cool one or two headings. There are many disadvantages including high maintenance costs due to arduous operating environment and difficult maintenance access as well as logistics problems (e.g. supply of 1.5 to 2.5 l/s of potable make-up water)

Some examples of Australian mine air cooling plants are shown in the following illustrations:

600kW Refrigeration Set for Underground Spot Cooler (WMC Olympic Dam)
Cooling and Heat Rejection Towers Not Shown







Chilled water is reticulated U/G via insulated pipes to underground cooling towers



2.4 Mine Gases

There are a surprisingly wide variety of gasses that can be found in underground mines, abandoned workings and caves. Some of these gasses may be poisonous, irritant, asphyxiant, radioactive or explosive and are hence of particular interest from a ventilation design viewpoint.

Mine ventilation systems are required to dilute and remove atmospheric contaminants caused by mining operations.

2.4.1 Constituent Gases of the Atmosphere

The atmosphere (fresh air) surrounding the earth surface contains a number of different gases combined as a mixture.

Composition of Air

GAS	VOLUME (%)
Oxygen (O ₂)	20.93
Nitrogen (N ₂)	78.11
Carbon Dioxide (CO ₂)	0.03
Minor Inert Gases	0.93
Water Vapour	Variable

Because this air is a mechanical mixture, it is possible to separate and identify each of the gases in the mixture. Any other substance or variation of these gases contained in the atmosphere are contaminants and are subject to Exposure Standards under legislation.

Any additional gasses or variation in the proportions of the gases normally found in the atmosphere are regarded as contaminants. Some characteristics of the more commonly found mine gases are discussed below.

2.4.2 Carbon Dioxide (CO₂)

Carbon dioxide is colourless, has a pungent or acrid smell and a “soda water” taste. It has a specific gravity relative to air of 1.53 (significantly heavier than air) and will not support combustion. It doesn't liquefy but will form dry ice at -78°C .

CO₂ has a TWA exposure limit of 5,000 ppm and a STEL of 30,000 ppm.

It can be found in coal mines and in mines hosted in carbonaceous rocks, such as limestone. It is also produced by diesel engines. The gas is about 20 times more soluble than Oxygen and diffuses rapidly into the bloodstream. The most noticeable effect of the gas is to cause the respiration rate to increase, which serves to alert the miner to the presence of the gas.

Physiological Effect of Carbon Dioxide

Percentage in Air	Effects
0.03	None (Normal atmospheric concentration)
0.5	Respiration increases by 5%
2.0	Respiration increases by 50%
3.0	Respiration increases by 200%.
5 – 10	Violent panting, leading to fatigue. Headache
10 – 15	Intolerable panting, severe headache, rapid exhaustion and collapse

2.4.3 Oxygen (O₂)

Oxygen is a colourless, odourless, tasteless gas, with a specific gravity relative to air of 1.1. Oxygen is the only gas whose concentration should be maintained *above* a recommended value. Oxygen depletion is caused by oxidation of minerals (e.g. Sulphides and Coal). Depletion of Oxygen also results from combustion (e.g. diesel engines, blasting etc). A deficiency of Oxygen implies greater than normal atmospheric concentrations of other gasses (even inert gasses). The physiological effect of various levels of Oxygen concentrations is shown below.

Concentration (by Vol.) in Air %	Typical Physiological Effects (Vary with individuals and period of exposure)	
20.93	Normal content of atmospheric air	
17.0 to 20.0	Lowest allowable concentration (variable in mine legislation)	
17.0	Noticably faster and deeper breathing rate (equivalent to 1,500m ASL elevation). Candle will not burn below 16%.	A safety lamp flame will go out at some point in this range.
15.0	Dizziness, buzzing in ears, rapid heart beat	
13.0	Work is difficult. Breathing becomes rapid and lips become blue. Nausea and headache develop slowly and may become very severe. May lose consciousness if exposure prolonged.	Dangerous for exposures over half to one hour.
10.0	Liable to faint and become unconscious.	Exertion leads to unconsciousness
9.0	Fainting, unconsciousness.	
7.0	Life endangered	
6.0	Convulsive movement, probable death.	
< 6.0	Rapid unconsciousness and death	

Note: If the oxygen content falls, the rate of breathing tends to increase to maintain the oxygen intake required. This automatic adjustment by the body to oxygen deficiency ends at about 17% oxygen and mental processes begin to become impaired although of course humans are not aware that they are being affected.

2.4.4 Carbon Monoxide (CO)

Carbon Monoxide has a high toxicity. Because of this and the fact that it is colourless, odourless and tasteless, it is an extremely dangerous gas. Carbon Monoxide has a specific gravity relative to air of 0.97 (almost exactly the same as air). It is flammable at concentrations of between 12.5 and 74% in air. The compound in red blood cells which transports Oxygen (haemoglobin) has an affinity for CO which is about 300 times greater than that for O₂. To further exacerbate the problem, the resulting compound (Carboxyhaemoglobin - CO-Hb) is significantly more stable than that formed by a combination of Oxygen and Haemoglobin (Oxyhaemoglobin) and does not readily decompose. The result of Carbon Monoxide exposure is that the red blood cells have a reduced ability to transport Oxygen. The effect is cumulative ie. with each breath, more and more CO is taken up by the blood, therefore reducing the concentration of O₂.

In severe poisoning, the effect may be very sudden and the first warning may be lack of response of the limbs, making escape difficult.

CO has a TWA exposure limit of 50 ppm and an STEL of 400 ppm.

Effects of Carbon Monoxide in the Blood Stream

Blood Saturation (% CO-Hb)	Effect
0-10	None
10-20	Possible headache
20-30	Headache, dizziness
30-40	Severe headache, weakness, nausea, loss of judgement, dimming of vision Possible collapse
40-50	Collapse
50-60	Collapse at rest, increased lung ventilation and pulse, convulsions
60-70	Convulsions, coma, depressed lung ventilation and pulse, disturbed judgement
>70	Slow weak pulse, respiratory failure and death.

Carbon monoxide is produced by fires, the oxidation processes (e.g. blasting) and sometimes issues from rock strata (especially in coal mines). It is also a component of combustion engine exhaust emissions.

2.4.5 Oxides of Nitrogen (NO_x)

Oxides of Nitrogen covers a mixture of gases usually found together. The most important of these are Nitric Oxide (NO) and Nitrogen dioxide (NO₂), both of which are classified as toxic. The proportion of NO is usually small and NO also readily converts to NO₂ in the presence of air and water vapour. Consequently, NO₂ is the oxide of most interest. This gas is brown in colour and dissolves readily in water to form Nitrous (HNO₂) and Nitric (HNO₃) acids. In sufficient concentration, these acids cause irritation and corrosion of the respiratory system and eyes. The associated bleeding and accumulation of fluid in the lungs can culminate in death from pulmonary oedema (flooding of the lungs). This can occur up to 24 hours after exposure, even after an apparently early recovery.

Oxides of Nitrogen are primarily produced by internal combustion engines and are also a significant constituent of blast fumes.

The TWA exposure limit for NO is 25 ppm and that for NO₂ is 3 ppm. As yet an STEL for NO has not been set, but for NO₂ the STEL is 5 ppm.

Physiological Effects of NO₂

Concentration ppm	Typical Physiological Effects for NO ₂
3	TWA
50	Moderately irritating to eyes and nose
100	Irritant to respiratory passages and to the eyes. (Headache, tightness of the chest or perhaps pain in chest and a cough). Dangerous if exposed for ½ to 1 hour.
200	Breathed for 20 minutes may cause collapse, this may be delayed for several hours.
250	Severe pulmonary oedema, probably fatal.

2.4.6 Sulphur Dioxide (SO₂)

In massive sulphide orebodies there can be spontaneous oxidation and heating resulting in sulphur dioxide being released to the ventilating air.

Sulphur dioxide is colourless and even at relatively low concentrations has a pungent, suffocating sulphurous odour, and acidic taste, making it readily detectable. It is highly toxic. It has specific gravity relative to air of 2.26 and is soluble in water, forming sulphurous acid. It is incombustible and is also not flammable. It is associated with oxidation of reactive sulphide ores. Of particular importance is the fact, that sulphur dioxide is produced by sulphide dust explosions. It is sometimes also noticed in mines that have their intake airways close to smelter stacks.

SO₂ exposure limits are a TWA of 2 ppm and an STEL of 5 ppm.

Effects of Sulphur Dioxide

Concentration (ppm)	Effect
3	Detectable by its odour.
100	Irritating to eyes and nose, uncomfortable to breathe.
500	Dangerous to life after only short exposures.

2.4.7 Hydrogen Sulphide (H₂S)

Hydrogen Sulphide is colourless and is readily detected in small concentrations by its unpleasant rotten eggs odour. Unfortunately, continued exposure to the gas (even for relatively short periods of time) leads to paralysis of the olfactory nerves, meaning that the sense of smell cannot thereafter be relied upon. The gas has a specific gravity relative to air of 1.19 and burns in air (in concentrations ranging from 4.5% to 45%), with a bright blue flame producing Sulphur Dioxide (SO₂).

Acidic action or effects of heating on Sulphide ores produce H₂S. It is also formed as a result of the decomposition of organic compounds. It is sometimes noticed near stagnant pools of water underground. Hydrogen Sulphide is sometimes associated with natural gas and oil reservoirs and can migrate through strata in solution.

H₂S has a TWA of 10 ppm and a STEL of 15 ppm. It is often associated with methane.

Hydrogen Sulphide Effects

Concentration (ppm)	Effect
0.1 – 1	Detectable by smell
100	Irritation to eyes and respiratory tract.
200	Intense irritation of eyes and throat.
500	After 30 mins serious inflammation of eyes and throat, coughing, palpitation, fainting, cold sweats.
600	Serious effects after a few minutes, bronchitis and chest pain.
700	Depression, stupor, unconsciousness, and death.
1,000	Paralysis of respiratory system and death.

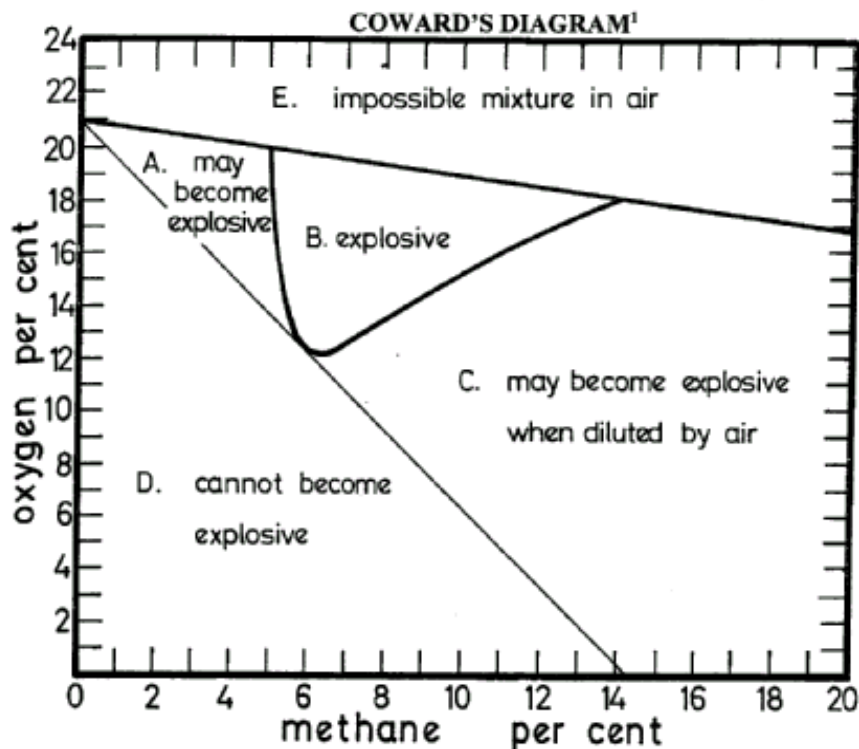
2.4.8 Methane (CH₄)

Methane is colourless, odourless and non-toxic. It has a specific gravity relative to air of only 0.55 and as a result, tends to layer against the backs in areas of low air velocity (laminar flow

conditions). Methane is of course very dangerous in mines because it forms an explosive mixture with air at concentrations of between 5 and 15 % (it produces an explosion of greatest force at a concentration of 9%). It is found in all coalmines, and has been responsible for many, many thousands of coal mining deaths directly as a result of methane explosions, or after the explosion as a result of Carbon Monoxide poisoning (CO is a product of incomplete methane oxidation).

Methane is often associated with other flammable/ explosive hydrocarbon gasses, predominantly hydrogen. The presence of methane in coal mines results from chemical and bacterial action on organic material. Methane is also surprisingly prevalent in metalliferous mining. It is often noted during diamond drilling. The most dangerous situations arise in metalliferous mining where large amounts of methane accumulate (e.g. the stope backs after firing).

Coward's Diagram shows the relationship between methane and oxygen concentration and explosibility.



¹ From "Subsurface Ventilation and Environmental Engineering" by Malcom J McPherson, 1993. Published by Chapman and Hall, 1993.

2.4.9 Coal Damps

Damp is an old miners term for gaseous products formed in coal mines to distinguish them from pure air. Although still in use they are not commonly used in today's mining.

2.4.9.1 Fire Damp

A combustible gas formed by the decomposition or distillation of coal or other carbonaceous matter. Consisting mainly of **Methane**. (Distillation is the heating of a substance in an atmosphere low in oxygen. This prevents oxidation if the heating was to take place in fresh air.)

Usually lighter than air and can accumulate in unventilated mine workings. A change in barometric pressure may cause this to be released to the ventilation system. Produced during mine fires by the distillation of coal.

Sometimes simply referred to as “gas”.

2.4.9.2 Black Damp

An atmosphere depleted of oxygen. More specifically an atmosphere containing variable mixtures of **Carbon Dioxide** (CO₂) and nitrogen (N₂), generally caused by the oxidation of carbonaceous material or coal and A typical mixture would be 15% CO₂ and 85% N₂. In its simplest form is the name given to CO₂.

Basically Black Damp is an extinctive atmosphere, hence the term “black”.

Usually heavier than air and can accumulate in unventilated mine workings. A change in barometric pressure may cause this to be released to the ventilation system.

2.4.9.3 Choke Damp

A mine atmosphere that causes ‘choking’ or suffocation due to insufficient oxygen. Could be any combination of CO₂ and CH₄ or other gasses or products of fires, smoke included, that may replace the oxygen content of the air.

In some places the name given to black damp.

2.4.9.4 Illawarra Bottom Gas

Any mixture of CO₂ and CH₄ ranging from almost 100% CO₂ to almost 100% CH₄ and a little N₂ that is capable of forming a flammable or explosive mixture when mixed with air.

Because of its density it tends to layer at the bottom of the drive, hence its name “bottom”.

2.4.9.5 After Damp

That mixture of gasses remaining after, a fire or explosion. (Some times referred to as “after gasses”.

2.4.9.6 White Damp

The term applied to carbon monoxide or more specifically atmospheres containing lethal quantities of CO.

2.4.9.7 Stink Damp

Atmospheres containing hydrogen sulphide with the odour the predominate factor. It is worthy to note that when concentrations exceed 50 ppm the sense of smell may be affected and the odour becomes undetectable.

2.4.9.8 Fire Stink

This is the smell indicating a spontaneous combustion. The odour is often associated with benzene.

2.4.9.9 Water Gas

A combustible mixture of gases with a typical composition being 45% each of CO and hydrogen with smaller amounts of CO₂, CH₄, N₂ and oxygen or after damp. Formed when water is hosed onto incandescent masses of coal when extinguishing the fire. The water gas produced could produce a secondary explosion.

Considered poisonous because of its high concentration of CO.

2.4.9.10 Producer Gas

A combustible mixture of gasses formed commercially by the action of air passing through a layer of incandescent fuel (coal, coke or charcoal). A typical mixture would be 10% CO₂, 15% CO, 74% N₂ and up to 1% of other gasses including CH₄.

An identical mixture may be formed in mine fires.

2.4.10 Ammonia (NH₃)

Ammonia is colourless, and has a very distinctive, pungent odour (the smell is familiar to those who have used certain disinfectant and window cleaning products). It has a specific gravity relative to air of only 0.65 and as a result, tends to layer against the backs in areas of low air velocity (laminar flow conditions). Ammonia is irritating or corrosive to exposed tissue, especially the eyes and the upper respiratory system. Inhalation of ammonia vapours may result in pulmonary oedema (flooding of the lungs) and chemical pneumonitis. Depending on the concentration, symptoms such as burning sensations, coughing, wheezing, shortness of breath, headache, nausea and eventual collapse may be experienced.

NH₃ has a TWA of 25 ppm and a STEL of 35 ppm

Ammonia readily passes into and out of solution in water. There are two main sources of ammonia in mines:

- As a refrigerant gas in mine cooling plants
- Chemical reaction involving ANFO explosive, cement and water.

Because of the toxic and irritant effects of ammonia, it is not used as a refrigerant in cooling plants where there is a possibility of the gas leaking into the mine atmosphere (although it is otherwise a particularly suitable gas for the purpose). The main source of ammonia in mines is from a chemical reaction involving ammonium nitrate, cement and water. The ammonium nitrate is sourced from spilt, or un-detonated ANFO. The cement source is generally shot-crete rebound (an increasing problem with more shot-crete usage). The reaction is as follows:

Calcium Oxide (a component of the cement) reacts with water to produce an alkali – Calcium Hydroxide: $\text{CaO} + \text{H}_2\text{O} \rightarrow \text{Ca(OH)}_2$

Next, the ammonium nitrate from the ANFO reacts with Calcium Hydroxide to produce ammonia gas (2NH₃), Ca(NO₃)₂ and water:



There have been an increasing number of ammonia “fumings”. Worst affected personnel seem to be those on charge-up (working at height, working close to face in poor ventilation, fresh ANFO “blow-back”, working shortly after shot-crete applied). The only solutions are to remove or isolate one or more of the components of the chemical reaction (i.e. water, ANFO or cement).

2.4.11 Radon (Rn) and Radon Daughters

Radon is a colourless and odourless gas. It is one of the isotopes¹⁰ produced by the radioactive decay of uranium to lead.

¹⁰ Naturally occurring elements comprise a mixture of isotopes. An isotope may have the same atomic number (number of protons in the nucleus of the atom), but different masses. For example, U238 has 92 protons and 146 neutrons, whereas the isotope U235 has the same number of protons (92), but only 143 neutrons.

Decay Series - Uranium to Lead (NOTE THAT THE ISOTOPES IN **BOLD** ARE RADON DAUGHTERS)

Nuclide	Radiation	Half-life
Uranium 238	α	4.5 billion yrs
Thorium 234	β	24 days
Protactinium 234	γ	
Protactinium 234	β	1.2 minutes
Uranium 234	α	250,000 yrs
Thorium 230	α	80,000 yrs
Radium 226	α	1,600 yrs
Radon 222 (gas)	γ	
Radon 222 (gas)	α	3.8 days
Polonium 218 (RaA)	α	3 minutes
Lead 214 (RaB)	β	27 minutes
Lead 214 (RaB)	γ	
Bismuth 214 (RaC)	β	20 minutes
Bismuth 214 (RaC)	γ	
Polonium 214 (RaC')	α	160 μ seconds
Lead 210	β	22 yrs
Lead 210	γ	
Bismuth 210	β	5 days
Polonium 210	α	140 days
Lead 206		infinite, stable

The half-life is the time taken for one half of the atoms in a radioactive substance to decay. During the decay process, various forms of radiation are released including Alpha (α) and Beta (β) particles as well as gamma (γ) radiation. Alpha particles are low energy, positively charged particles that can travel a few centimetres in air. The respiratory system can be damaged by inhaled alpha particles, but alpha particles do not penetrate the skin surface. Beta particles are electrons. They can penetrate the skin and cause damage to the body's cells and organs. Gamma radiation is electromagnetic radiation that can penetrate deeply into the body.

The half-life of radon is relatively long (at 3.8 days). As a result, it doesn't tend to expose the lungs to a significant amount of radiation energy. It is a different story with the relatively short-lived radon daughters (referred to radon progeny in these more politically correct times)! Inhalation of radon daughters in sufficient concentration over a long enough period of time will increase the likelihood of exposed personnel developing lung cancer. Radon daughters tend to attach themselves to surfaces such as dust particles and aerosols. As a result, the control of ambient dust (especially respirable dust) and diesel soot levels in uranium mines is of particular importance. It is also important in order to reduce the exposure to alpha emitting dust particles. When all else fails, mandatory respiratory protection is sometimes used as a method to limit radon daughter exposure.

In underground uranium mines, radon daughters and radioactive dust particles are just some of the radiation hazards. The uranium ore emanates gamma radiation, which also contributes to the total radiation exposure of the workforce. In contrast to dust and radon daughters, Gamma radiation exposure can't be controlled by ventilation. Instead, it is managed by physically

shielding the workforce from the ore (particularly high grade ore) and by mine design and shift rotation to limit the time spent in areas where the gamma radiation levels are high.

At one, low grade underground uranium mine, the approximate contributions to the total radiation dose are:

- Radon daughters 40%
- Gamma Radiation 40%
- Airborne alpha emitting dust 20%

Assuming the ventilation system is well designed and managed, the relative contribution to total dose from gamma radiation will increase for mines with higher uranium grades.

It is important to note that the concept of *residence time* is of critical importance when designing ventilation systems for uranium mines. Once the radon emanates from the rock, it becomes a “time bomb”, and begins immediately to disintegrate into radon daughters. The aim of the ventilation system design is to remove the radon gas from the underground environment as quickly as possible after it is produced. Depending on the ore grade, ventilation design criteria can specify underground air residence times (after exposure to rock containing uranium) of less than 10 to 15 minutes.

Areas of low velocity, with high ore rock surface exposure (e.g. stopes and sealed off workings) are radon daughter “breeding grounds”. Personnel should not be exposed to the resulting high radon daughter concentrations in the air leaking or exhausted from these locations. This contaminated air should flow to a surface exhaust airway via as direct a route as possible. The ventilation system design should ensure that personnel work in “fresh” air that has travelled from the surface to the working place via a direct route in low-grade (waste) rock. In order to limit exposure of personnel to radon daughters, considerable care with mine design and scheduling is required in uranium mines and accordingly, ventilation considerations are often the most important mine design criterion.

The source of radon in mines is primarily via emanation from rock containing uranium and radium. The gas is found in underground mines (not just uranium mines) and sometimes also in caves (many of which have poor ventilation and therefore long residence times).

Determination of a “safe” or “acceptable” exposure standard for radon and radon daughters is complex. Exposure to these gasses constitutes just one of several contributors to the total radiation exposure. The total effective radiation dose is measured in units of Sieverts (Sv). The Sievert is a unit of radiation derived health risk. The prevailing annual limit for radiation workers is 20 mSv/ year, averaged over 5 yrs. Note that practice of the “ALARA” (As Low As is Reasonably Achievable) principle is always the recommended approach with respect to radiation doses and a number of operations have adopted internal standards which aim to achieve dosages that are less than a third or a quarter of the prevailing limits.

SUMMARY OF GASSES

Name of Gas	Symbol	Properties	Smell	Flammable Limits	TWA		STEL		Physiological Effects
					(ppm)	(%)	(ppm)	(%)	
Oxygen	O ₂	Colourless Tasteless	Nil	In contact with oil or grease	-	17.0 (19.0 in mines)	-	13.0	Essential to maintain life
Carbon Monoxide	CO	Colourless Tasteless	Nil	12.5% - 74%	50	0.005	400	0.04	Displaces oxygen in blood 200 ppm Drowsiness; Headache after 2 hours work 400 ppm Headache after 45 mins work 1200 ppm Palpitations after 10 mins work 2000 ppm Unconsciousness after 10 mins work >3000 ppm Death possible
Carbon Dioxide	CO ₂	Colourless Soda taste	Slight pungent	Non-flammable	5000	0.5	30000	3.0	Increased respiration Depression of breathing
Hydrogen	H ₂	Colourless Tasteless	Nil	4.0% - 74%	-	-	-	-	Non-poisonous Asphyxiant
Hydrogen Sulphide	H ₂ S	Colourless Sweet taste	Rotten Eggs	4.5% - 45%	10	0.001	15	0.0015	100 ppm Irritation to eyes and throat : Headache 1000 ppm Immediate unconsciousness
Nitrogen	N ₂	Colourless Tasteless	Nil	Non-flammable	-	-	-	-	Non-poisonous Asphyxiant
Oxides of Nitrogen	NO _x		Firing fumes		25	0.0	-	-	100 ppm irritant to respiratory passage and eyes 200 ppm After 20 mins may cause collapse usually delayed 400 ppm after 15 mins exposure may be fatal
Nitrogen Dioxide	NO ₂	Brown colour Acrid taste	Firing fumes Acrid	Non-flammable	3	0.0003	5	0.0005	50 ppm Irritating to eyes and nose 250 ppm severe pulmonary oedema collapse is usually delayed
Methane	CH ₄	Colourless Tasteless	Nil	5.0% - 14.0%	Stop diesel engines when = or > 1.0% Turn off electrical power when = or > 1.25% Remove people when = or > 2.5%				Non-poisonous Asphyxiant
Sulphur Dioxide	SO ₂	Colourless Acid taste	Burning Sulphur Pungent Suffocating	Non-flammable	2	0.0002	5	0.0005	Oedema

2.5 Diesel Engines

Dr. Rudolf Diesel first patented the device and principles for a compression engine in Germany in 1892. The diesel engine as it is now known, is used almost exclusively as the engine of choice in Australian Mines. The exception being some electric LHD's and trucks. Other fuels such as hydrogen have been investigated since the 1960's and there is one hydrogen engine operating in Canada today but generally speaking their use remains in the laboratories until all the safety implications have been overcome.

Diesel engines are considered to be reliable, robust and relatively easy to maintain, and particularly efficient at partial loads. These engines do not rely on spark plugs for fuel ignition but rely on the in-cylinder temperature, generated on the compression stroke, for ignition of the injected fuel. For this to occur, air is injected into the cylinder and compressed to a high pressure with a corresponding temperature rise. As the cylinder nears the top of its stroke the temperature of the gases rise to temperatures in excess of 540°C, well above the ignition point for diesel fuel. A fine mist of fuel is injected into the gap where it combusts. The resulting physical and chemical processes lead to auto-ignition just prior to the cylinder reaching top dead centre, the combustion energy released forces the piston down to bottom dead centre in the power stroke.

Engines used in mining can be broadly classified as, either direct injection (DI), indirect injection (IDI or otherwise referred to as pre combustion PC) they can be naturally aspirated or turbo charged for increased power and performance. PC engines are considered to be more acceptable for use in underground workings because they generally have lower gas emissions.

Electronic fuel management systems are progressively being introduced more extensively into underground hard rock mines, but it is worth a note that electronic fuel management systems have yet to be introduced into underground coal mines and this will only occur once they are able to provide "intrinsically safe" mechanisms. Although this fuel management system has helped reduce the more visible DPM the side effect has been the increase (albeit minor) in levels of oxides of nitrogen (NO_x).

2.5.1 Diesel Exhaust Emissions (DEE)

The production and concentrations of gases in DEE is dependent upon

- Engine type and manufacturer
- Engine speed
- Engine adjustment and maintenance
- Working load of the engine
- Type of fuel

Because of these variables it is extremely difficult to provide absolute values of the quantity and concentrations of DEE gases.

Diesel fuels consist primarily of Carbon (84.5%) and Hydrogen (15%) with a small amount of Sulphur (0.5%). Typically diesel fuels contain two hydrogen atoms for each carbon atom and can therefore be represented as C₁₂H₂₄. Complete combustion of 1kg of diesel fuel would result in Carbon Dioxide (CO₂) water vapour (H₂O) and Sulphur Dioxide (SO₂) in the following proportions

$$\text{CO}_2 = 0.845 \times \frac{44}{12} = 3.10\text{kg}$$

$$\text{H}_2\text{O} = 0.150 \times \frac{18}{2} = 1.35\text{kg}$$

$$\text{SO}_2 = 0.005 \times \frac{64}{32} = 0.01\text{kg}$$

A total of 4.46kg of gas.

For complete combustion the amount of Oxygen required is $4.46 - 1.00 = 3.46\text{kg}$ and as standard air contains 23.15% Oxygen, by mass, the air required for complete combustion is $\frac{3.46}{0.2315} = 14.95\text{kg}$. In other words an air to fuel ratio of 1:14.95, or alternatively each 0.0669kg of

fuel requires 1.0kg of air for complete combustion. Fuel to air ratios in normal conditions are 0.01kg at idle and 0.05kg at full throttle up to a maximum of 0.06kg. At full throttle full load this could be as high as 0.08kg but at this ratio there is a large amount of unburnt fuel emitted. When averaged over a full shift the fuel to air ratio would typically be between 22:1 (0.045kg) and 25:1 (0.04kg fuel)

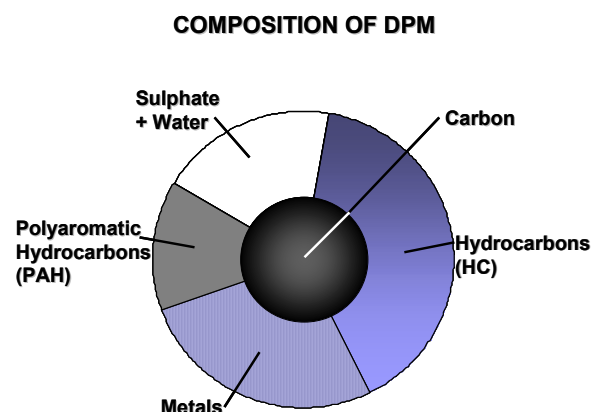
In reality diesel engines never operate at 100% efficiency, seldom at full load and consequently complete combustion is never achieved. The products of this incomplete combustion are Carbon Monoxide (CO), Hydrocarbons (including aldehydes) – (HC), Carbon (soot defined as diesel particulate matter) – (DPM), Oxides of Nitrogen (NO_x) including Nitric Oxide (NO) and Nitrogen dioxide (NO₂). Other toxic substances like polyaromatic hydrocarbons (PAH) are also found in both the HC and DPM component of the DEE. The concentrations of gasses in DEE are directly related to the quantity of fuel used.

Fuel consumption is related to work load on the engine (i.e. the higher the work load the greater the fuel consumption). Maximum fuel consumption is achieved when the engine is operating at “torque stall”, therefore maximum concentrations and emission levels of DEE gases. In the case of mining equipment this would be when a LHD unit is bogging and when a truck is hauling fully loaded up an incline. Ideally exhaust gas emission rates should be obtained for all normal operating situations. For example a LHD cycle would include loading, hauling full, dumping full, hauling empty and Idle.

Although the liquid fuel droplets in the injected mixture begin to decompose on ignition the fuel rich zones associated with them result in incomplete combustion and the formation of CO rather than CO₂ and as the load increases the amount of excess Oxygen decreases and tends to increase the formation of CO. Because there is a large amount of Nitrogen in the injected air used for combustion this along with the pressure and temperature of combustion some of the nitrogen is oxidised to NO and this then oxidises at a much slower rate to NO₂ to the extent that only 10% is oxidised by the time it is exhausted from the engine.

2.5.2 Diesel Particulates

Diesel particulate matter (DPM) is the soot particles emitted with diesel engine exhaust. The size of the particles is almost totally within the respirable range. The particles contain hundreds of adsorbed compounds, some of which are known to be carcinogenic. DPM is thought to be a “potential carcinogen” but no unequivocal evidence on this matter is currently available and the subject is as a result, still somewhat controversial.



DPM is a complex substance and has been to object of constant investigation for a number of years. More particularly since the formation of a task force initiated by Coal Services Pty Ltd (formally the Joint Coal Board) in 1997 to determine the management and control of DPM in Coal mines. As yet the hard rock industry has not embraced these studies but will be keeping a watchful eye on proceedings since the introduction of control legislation in the USA in 2001.

The American Conference on Governmental Industrial Hygienists (ACGIH) has proposed a general exposure standard for Diesel Particulates (DP) of 0.15 mg/m^3 , but has failed so far to achieve worldwide recognition. Whilst there are currently no diesel particulate exposure standards for mines in Australia, if we were to follow the USA lead then an exposure standard of 0.4 mg/m^3 (eight-hour exposure) will be introduced some time in the future. The USA control legislation is pointed at manufacturers and the reduction of DPM in the exhaust emissions. and a standard along these lines was introduced for the USA mining industry in 2001.

It is understood that establishing a diesel particulate exposure standard for underground mines in Australia is not on the short-term agenda. Adoption of the ALARA (As Low As Reasonably Achievable) principle to workforce diesel particulate exposures would however be a pragmatic and recommended response to the issue.

In recent years design strategies have concentrated on the achieving more complete combustion and minimising the formation of in-cylinder particulate matter. The formation of NO_x is the sole function of the available oxygen and the temperature, and the higher in-cylinder temperatures have subsequently increased the amounts of nitrogen that are oxidised to Nitric Oxide (NO), some of which is re-oxidised to Nitrogen Dioxide (NO_2).

Hence a dilemma, decreased DPM but at the expense of increased NO_x emissions.

The 1990's has seen some major advances in engine technology to reduce DEE.

- Fuel injector design have allowed manufacturers to control the rate of fuel injection resulting in lower emissions of DPM and NO_x
- Fuel injector pressure has been increased resulting in better atomisation of the fuel in the combustion chamber resulting in decreased DPM
- Turbo-charging has resulted in better combustion and decreased DPM whilst cooling the compressed air supplied to the intake manifold reduces the NO_x that would result from the increased combustion temperatures.
- Improved intake manifold and port configurations have achieved better in-cylinder air distribution eliminating fuel rich spots and inturn decreasing DPM and hydrocarbons emissions.
- Combustion chambers have been redesigned to achieve better mixing resulting in improved combustion and decreased DPM and hydrocarbons
- Oil control has been improved significantly as prior to 1990 as much as 30% of DPM was attributed to lubricating oil. This improvement has reduced DPM by as much as 10%.

A number of "exhaust conditioners" have been developed over the years. These conditioners are extremely expensive and require a high level of maintenance. Some, such as the CO catalytic converters have been highly successful whilst others are still in development stage. The obvious problem is to get a single conditioner for all contaminants as experience has shown that one problem is solved and another created.

Available methods for reducing underground DPM include:

- Use more efficient diesel engines (eg engines with electronic engine fuel-air mixture control systems)

- Pay close attention to engine maintenance (tuning, air cleaner maintenance etc)
- Use low Sulphur fuel (<0.05% Sulphur). This type of fuel is now becoming more readily available in Australia.
- Use exhaust particulate filters especially on production equipment where high exhaust temperatures can be maintained for long enough to burn trapped particulates off catalysed ceramic filters. Up to 85% of particulates can be captured using particulate filters.

A recommended reference is - "Diesel Emissions in Underground Mines – Management and Control" NSW Minerals Council, October 1999. It states, *"More research into the effects of ventilation and diesel particulate exposure is required before comprehensive guidance can be provided on this issue"* one can't help but agree.

2.5.3 Dilution of Diesel Exhaust Emissions (DEE)

The concentrations of contaminants in the ambient air are directly related to the total quantity of gases and particulates emitted from the diesel engine and the mine dilution ventilating air. If the engine operating mode, emission characteristics, and the engine exhaust air volume are known, the quantity of emissions from the engine can be calculated.

Studies have shown that diesel engines in operating mines have a relatively low effective load factor, some times referred to as the duty cycle, and could be as little as 20% to 40% of full load over the entire shift.

One of the questions asked of ventilation engineers is the quantity of ventilating airflow necessary to dilute these contaminants below statutory exposure limits.

Most Australian state legislation provides some guideline for this dilution rate and in Western Australian legislation this is very specific. It is good management to check this legislation for the state in which the mine is located.

One equation used to calculate this dilution factor is written as

$$Q = \frac{Q_g}{A_C - N_C} - Q_g \quad \text{Equation 2-4 Dilution Equation}$$

Where

Q = Quantity of airflow required for dilution (m^3/s)

Q_g = Exhaust gas Emission Rate (EGER) (m^3/s per kW)

A_C = Maximum allowable gas concentration in the workplace. (m^3/s)

N_C = the normal concentration of gas present in the diluting air (m^3/s)

Notes: $\text{m}^3/\text{s} = \text{ppm} / 1,000,000$

EGER = Measured concentration x volume of exhaust gasses.

Based on the NOHSC 8 hour exposure standard of 30ppm for CO, a text book exhaust emission rate of $0.0015 \text{ m}^3/\text{s}$ per kW of installed power and an undiluted concentration of CO of 1500 ppm in the exhaust (i.e. the maximum allowable limit) the dilution airflow required for DEE is $0.075 \text{ m}^3/\text{s}$ per kW.

$$Q = \frac{0.0015 \times 0.0015}{0.00003 - 0.0} - 0.0015 \times 0.0015$$

$$= 0.07499 \text{ m}^3/\text{s}$$

The US Bureau of Mines (USBM) recommends the use of the following simplified formula to calculate the dilution factor for DEE.

$$Q = \frac{VC}{Y} \quad \text{Equation 2-5 DEE Dilution (USBM)}$$

Where

Q = Quantity of airflow required for dilution (m³/s)

V = Volume of gas produced (m³/s)

C = Concentration of gas in the exhaust (ppm)

Y = Maximum allowable concentration in the workplace (ppm)

$$\begin{aligned} \text{i.e. } Q &= \frac{0.0015 \times 1500}{30} \\ &= 0.07500 \text{ m}^3/\text{s per kW} \end{aligned}$$

Similarly for NO_x. TLV = 25ppm eight-hour exposure

$$\begin{aligned} Q &= \frac{0.0015 \times 1000}{25} \\ &= 0.0600 \text{ m}^3/\text{s per kW} \end{aligned}$$

From this calculation the minimum dilution airflow should be 0.075m³/s per kW of installed diesel power.

The use of these equations is fine but only IF the emission rate is known. We can easily determine, by measurement the concentration of the contaminants whereas the rate of emission can be calculated from manufacturers data.

Emission rate is a function of the number and dimensions of the cylinders and the speed of the motor. Knowing these the rate at which the gasses are forced from the engine enables the correct calculation of the rate of emission, obviously, the greater the speed of the engine the greater the rate of emission.

For example consider the engine data provide by a engine manufacturer shown below.

Engine:	Inline 6 cylinder 4 stroke	
Power Rating:	475 HP (354kW) @ 2100 RPM	
Max Torque:	1550 ft.lb (2101 Nm) @ 1200 RPM	
	Rated Speed	Max Torque Speed
Exhaust Temperature:	432 degC	518 degC
Stroke:	139mm	
Bore:	130mm	

From this data we can estimate the exhaust flow (emission) rate for the engine

Calculation of exhaust swept volume

$$D = \left(\frac{b}{2} \right)^2 \times \pi \times S \times \frac{n}{10^6} \quad \text{Equation 2-6 Engine Displacement Volume}$$

Where: D = Engine Displacement Volume (litres)

b = Bore diameter (mm)



$$\pi = 3.1428$$

S = Stroke length (mm)

n = Number of Cylinders

Therefore

$$D = \left(\frac{130}{2} \right)^2 \times \pi \times 139 \times \frac{6}{10^6}$$

$$= 11.1(\text{litres})$$

$$SEV = \frac{D \times s}{2000} \quad \text{Equation 2-7 Swept Exhaust Volume (Four stroke Engine)}$$

Where: SEV = Swept Exhaust Volume (m³/min)

D = Engine displacement (litres)

s = Engine speed (rpm)

2000 = Four Stroke engine factor

$$SEV = \frac{D \times s}{1000} \quad \text{Equation 2-8 Swept Exhaust Volume (Two stroke Engine)}$$

Where: 1000 = Two Stroke engine factor

The engine data provided is a four-stroke therefore

at the rated speed Swept Exhaust Volume = $\frac{11.1 \times 2100}{2000}$

$$= 11.7 \text{ m}^3/\text{min}$$

and at max torque speed Swept Exhaust Volume = $\frac{11.1 \times 1200}{2000}$

$$= 6.7 \text{ m}^3/\text{min}$$

Correcting for temperature and assuming pressure (P) is constant

$$\frac{P_1 V_1}{T_1} = \frac{P_2 V_2}{T_2} \quad (\text{Universal Gas Law})$$

Where: T = temperature in °K

V = Swept Volume (m³)

at the rated speed

$$\frac{11.7}{25 + 273} = \frac{V_2}{432 + 273}$$

$$= 27.7 \text{ m}^3$$

and similarly for maximum torque

$$= 17.8 \text{ m}^3$$

A further correction is then made for turbo charging and after cooling. Typically this 1.5 but will vary from engine to engine.

Correcting for Turbo charging the emission rates now become 41.5 m³/min at the rated speed (i.e. 27.7 x 1.5 = 41.5) and, 26.7 m³/min at maximum torque (i.e. 17.8 x 1.5 = 26.7).

“I can remember when the air was clean and sex was dirty.”

George Burns

3 PROPERTIES OF AIR

Air is a mixture of a number gasses. Air is a perfectly mixed (homogeneous) mixture of gasses, in spite of the fact that the various component gasses have different densities.

Composition of Air

Oxygen	20.93%
Nitrogen	78.11%
Carbon Dioxide	0.03%
Inert Gases	0.93%
Water Vapour	Variable

Note that layering of gasses (e.g. methane, which is lighter than air and Carbon Dioxide, which is heavier than air) sometimes occurs in mine airways, where the airflow is very low. If these gasses seep slowly into an area of very low airflow (laminar flow conditions - $\ll 0.1$ m/s), a meniscus can form at the boundary between the gas and the air, leading to separation and layering. Providing a turbulent air can easily break up this layering. The gas will then mix evenly with the air, and will never separate again.

Mine air also contains a percentage of water vapour. The vapour is regarded as being a separate component or impurity (i.e. it is not viewed as being part of the air).

3.1 Air Temperature

Mine air temperatures are measured in SI units of Degrees Celsius. It is useful to recall that at normal sea level pressure, water freezes at 0°C and boils at 100°C .

The lowest (theoretically) possible temperature is -273.15°C (sometimes known as “absolute zero”). If we change the Celsius scale so that absolute zero (-273.15°C) is reassigned the value of “0” then we will have what is known as the Kelvin temperature scale (K). So $0\text{ K} = -273.15^{\circ}\text{C}$ and $273.15\text{ K} = 0^{\circ}\text{C}$. In other words,

$$T(\text{K}) = T(^{\circ}\text{C}) + 273.15$$

Equation 3-1 – Relationship Between $^{\circ}\text{C}$ and K

There are two types of air temperature readings that we deal with routinely in mining. They are dry bulb and wet bulb temperatures.

Dry-bulb temperature is simply the air temperature. When measuring dry-bulb air temperatures it is important to avoid the effect of radiative heat transfer (i.e. temperature readings should not be taken near surfaces whose temperatures differ by more than several degrees from the air temperature – e.g. avoid measuring air temperature near a hot diesel engine)

Wet-bulb temperature is measured with a thermometer whose bulb is covered with a thin, wet cotton sleeve. As with dry-bulb temperature readings, wet bulb air temperature measurement near surfaces whose temperatures differ by more than a few degrees from the dry-bulb air temperature, are avoided. The bulb must be aspirated by an airflow that is at least 3m/s, to ensure accurate reading. Due to the effect of evaporation, the wet bulb temperature will always be less than, or equal to the dry-bulb temperature.

3.2 Charles' Law

Charles' law states that the same rise of temperature produces in all gasses the same increase in volume, provided the pressure is kept constant.

In other words, at constant pressure:

$$\frac{T_1}{T_2} = \frac{V_1}{V_2} \quad \text{Equation 3-2 – Charles' Law}$$

Where: T = Absolute temperature (K)

V = Volume occupied by gas

3.3 Boyle's Law

Boyle's law states that the volume occupied by a gas is inversely proportional to the absolute pressure exerted upon it, provided the temperature is kept constant. In other words if the pressure on the gases increases it's volume will decrease. Conversely if the pressure decreases, the volume of the gas will increase:

$$P_1 V_1 = P_2 V_2 \quad \text{Equation 3-3 – Boyle's Law}$$

Where: P_1 = Initial absolute pressure

V_1 = Initial volume occupied by gas

P_2 = Final absolute pressure

V_2 = Final volume occupied by gas

3.4 Universal Gas Law

Charles' law and Boyle's Law can be combined mathematically to produce what is known as the Universal Gas Law:

$$\frac{P_1 V_1}{T_1} = \frac{P_2 V_2}{T_2} \quad \text{Equation 3-4 – Universal Gas Law}$$

Therefore, for a given quantity (mass) of a given gas, $\frac{PV}{T}$ is a constant. The constant for one kilogram of a given gas is known as R , hence $\frac{PV}{T} = R$.

Note that because we now specify that we have one kilogram of gas, then the volume (V) is in units of m^3/kg – i.e. specific volume.

R varies depending on the gas type.

For dry air, $R = 0.2871 \text{ kJ/kg}$, assuming pressure (P) is in kPa, temperature (T) is in K and specific volume (V) is in m^3/kg . Note that a Joule is $1 \text{ N}\cdot\text{m}$.

3.5 Density of Dry Air

The density of dry air can now be readily determined using the Universal Gas Law. An example calculation is outlined below:

Example

What is the density of dry air at a barometric pressure of 95 kPa and a dry bulb temperature of 25°C ?



Rearranging Equation 3.4

$$V = \frac{RT}{P} \text{ and substituting...}$$

$$V = \frac{0.2871 \times (25 + 273)}{95}$$

$$= 0.90 \text{ m}^3/\text{kg}$$

$$\text{Density } (\rho) \text{ kg/m}^3 = \frac{1}{V(\text{m}^3/\text{kg})}$$

$$= 1.11 \text{ kg/m}^3$$

This is a very useful “first pass” method to estimate the density of air. It is accurate to within $\pm 0.5\%$ for any condition between dry air at any temperature and saturated air (100% relative humidity) up to 25°C.

Another useful application of the Universal Gas Law is illustrated below:

Example

200 m³/s of dry air at a temperature of 30°C and barometric pressure of 95 kPa flows down an intake shaft into a mine. At the bottom of the shaft, the air temperature is measured as 37°C and the barometric pressure is 102 kPa. What is the flow rate at the base of the shaft if there is no leakage and no moisture pick-up in the shaft?

Rearranging Equation 3.5:

$$V_2 = \frac{P_1 V_1 T_2}{T_1 P_2}$$

And substituting:

$$V_2 = \frac{95 \times 200 \times (37 + 273)}{(30 + 273) \times 102}$$

$$V_2 = 191 \text{ m}^3/\text{s}$$

The result shows that we have “lost” 9 m³/s (4½%) of our airflow before it reaches the base of the shaft. The change in volume flow due to air compression is an important factor when designing and monitoring ventilation systems for deeper (e.g. >1,000m below surface) mines. In these deeper mines, it is therefore often more convenient to deal with air flow rates in terms of mass flow (i.e. kg/s). For the majority of mines however, the change of volume due to compression of the air is small enough to be ignored.

3.6 Moisture in Mine Air

Up to now, we have assumed “dry” air for all of our calculations. This is a simplistic assumption since in mine airways, the air invariably also contains some water vapour.

The amount of water vapour contained in a volume of air is dependent on the temperature of the air. At higher temperatures, the amount of water vapour that the air can contain increases. When air contains the maximum possible amount of vapour at a given temperature, it is said to be **saturated**. Saturated air has a relative humidity of 100%.

If air containing a certain amount of water vapour is cooled, a temperature will be reached where the air is unable to contain all of the water vapour. The vapour will condense out into water droplets. This temperature is known as the **dew point temperature**.

Condensation of water vapour can cause “fogging” and can sometimes be a problem in mine airways where warm, humid air is cooled below its dew point (e.g. by a mine air cooler). Condensation of water sometimes also occurs in upcast ventilation shafts – in this case the cooling of the mine air is caused by the reduction in air pressure as the air travels upwards (the opposite of autocompression). The condensed water droplets can cause problems with surface fan operation.

3.7 Density of Humid Air

The “Standard” density of dry air at 20°C and 101,325 Pa barometric pressure is **1.2 kg/m³**

Unless otherwise stated, the performance figures supplied with mine fans assume that the fans will operate at **standard** air density.

Unfortunately, many mine fans operate in locations where the fan inlet air density is significantly different to standard air density. It is therefore necessary to understand how to calculate the actual air density against which the fans operate. In many situations, Equation 3.4 is sufficiently accurate, however where improved accuracy is important, it is necessary to employ a calculation method that takes into account the effect of the moisture contained within the mine air.

The density of air is the mass per unit volume and varies according to the water vapour contained in the air and three input values are required to determine the density of air which contains water vapour. They are:

- Barometric Pressure
- Dry Bulb Temperature
- Wet Bulb Temperature.

The air density can be calculated using the relationship described as

$$\rho = \frac{P - 0.378e}{0.287045(273.15 + t_d)}, \text{ (kg of air/m}^3\text{)} \quad \text{Equation 3-5 Density of Humid Air}$$

Where

ρ = Air Density (kg/m³) (Some texts describe this as *vair* rather than air, this is to distinguish between dry air and air containing water vapour.)

P = Atmospheric Pressure (kPa)

t_w = Wet-bulb temperature(°C)

t_d = Dry-bulb temperature (°C)

e = Partial Pressure of water vapour (kPa)

$$e = \frac{e_{sw}(371.4 + 0.24t_d - 0.6t_w) - 0.24(t_d - t_w)P}{371.4 + 0.04t_d - 0.4t_w} \quad \text{Equation 3-6 Partial Pressure of Water Vapour}$$

e_{sw} = Saturated vapour pressure at the wet-bulb temperature (kPa)

$$e_{sw} = \exp. \frac{17.27t_w}{237.3 + t_w} \quad \text{Equation 3-7 Saturated Vapour Pressure at the Wet-bulb Temperature}$$

EXAMPLE

What is the density of the air in a drive where the wet and dry bulb temperatures are (respectively) 25 and 30°C and the barometric pressure is 97.5 kPa.

We will solve this problem using both the approximate method (Equation 3.4) and the more accurate method (Equation 3.5).

Approximate Method:

Rearranging Equation 3.4...

$$\rho = \frac{P}{RT} \text{ and substituting...}$$

$$\rho = \frac{97.5}{0.2871 \times (273 + 30)} \text{ kg/m}^3$$

$$\rho = 1.121 \text{ kg/m}^3$$

In fact this is the density of dry air.

More Accurate Method:

Calculate the Saturated vapour pressure at the wet-bulb temperature

$$e_{sw} = \exp. \frac{17.27 \times 25}{237.3 + 25}$$

$$= 3.166 \text{ kPa}$$

Calculate the Partial Pressure of water Vapour

$$e = \frac{3.166(371.4 + (0.24 \times 30) - (0.6 \times 25)) - 0.24(30 - 25)97.5}{371.4 + (0.04 \times 30) - (0.4 \times 25)}$$

$$= 2.852 \text{ kPa}$$

Use Equation 3.5 to calculate the air density (ρ):

$$\rho = \frac{97.5 - (0.378 \times 2.852)}{0.287045(273.15 + 30)}$$

$$\rho = 1.108 \text{ kg/m}^3$$

3.8 Air Pressure

Every person who works in an underground mine knows that the fan produces the airflow and generally accept this fact without needing (or wanting) to know any more. The operation, use and placement of fans will be discussed later in these notes. At this point we need to understand that fans create a pressure differential that causes the air to flow through the workings.

To assist with understanding consider the following, when we pump up a tyre the pressure in the tyre becomes higher than the pressure of the atmosphere. Similarly if we were to suck the air from a vessel (create a vacuum) the pressure of the air outside the vessel would be greater than the pressure of the air inside the vessel. When the tyre valve is opened the air flows from the tyre and if the vessel is opened the air will flow into the vessel. In both cases the air will **flow from the high-pressure area to the low-pressure area** until the pressure inside the vessel is the same as the pressure outside the vessel. When this happens the flow will stop.

This is also seen on a weather map where the wind blows from the HIGH pressure to the LOW pressure. The greater the pressure difference the faster the wind blows.

In ventilation (mines, buildings or otherwise) for air to flow from one point to another there must be a difference in pressure between the two points. This difference in pressure is known as the **ventilating pressure** and the following rules apply.

- (1) Air will always flow from a high pressure point to a low pressure point and as long as this pressure is maintained, air will continue to flow,
- (2) The larger the pressure difference between these two points the greater the quantity of air will flow. This assumes the resistance between the two points remains unchanged.
- (3) Resistance to pressure reduces the ventilating pressure, ie the pressure is used up overcoming resistance to the airflow.
- (4) If the pressure difference between two points remains the same and the resistance to airflow between these points is increased the air quantity will decrease

Pressure is the force applied per unit area and in ventilating terms is usually expressed in Pascals.

One Pascal (Pa) = One Newton per square metre (N/m^2)

There are many other expressions for pressure including inches of water gauge, millimetres of mercury all of which are required to be converted to Pascals for use in mine ventilation calculations. Of importance in to mine ventilation practitioners is the different types of pressure.

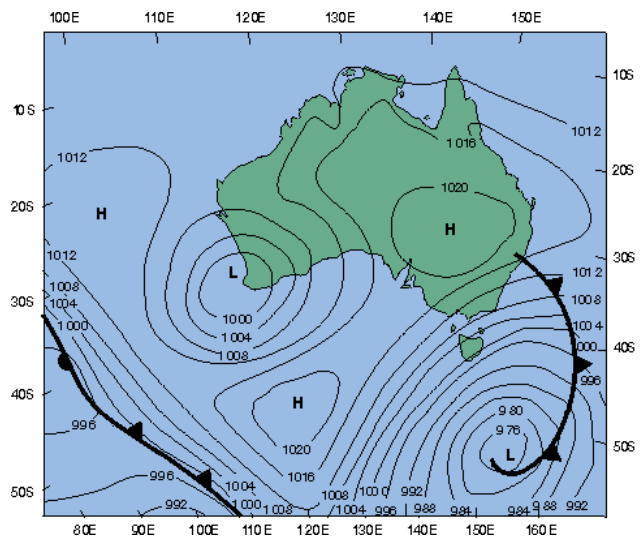
3.8.1 Atmospheric Pressure

The earth is contained in an atmosphere and by the force of gravity, it exerts a pressure on the surface of the earth caused by the weight of the air above.

Atmospheric pressure at sea level is equivalent to a mass of 10,000 kg on one square meter (m^2) of surface and is expressed in the following ways:

- 1 Atmosphere,
- 1.013 Bar,
- 101.325 kPa,
- 1013 mb,

The pressure shown on weather charts is usually expressed in millibars (mb) and often referred to as the Barometric Pressure.

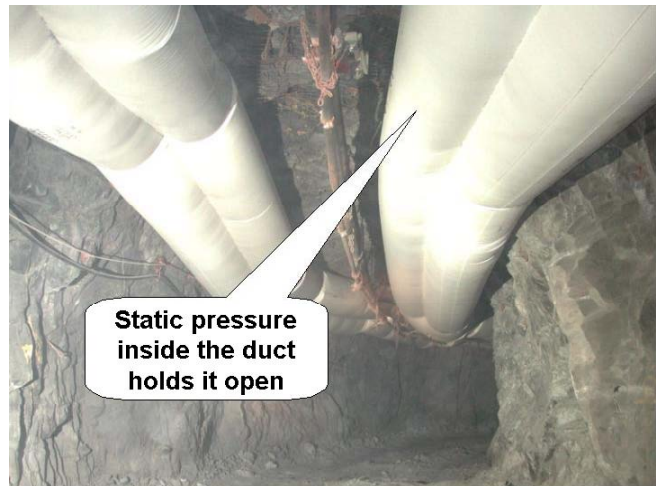
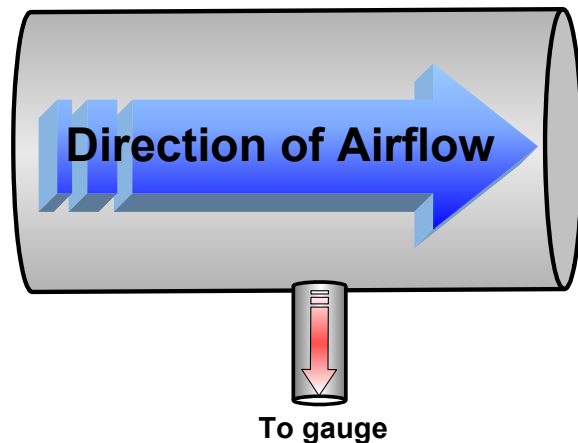


3.8.2 Barometric Pressure

Barometric pressure is simply a description of the instrument used to measure the pressure of the atmosphere in which it is situated. In mine workings the pressure measured using a barometer includes pressure changes caused by fans and therefore is not always be a true indication of the atmospheric pressure outside the mine.

3.8.3 Static Pressure (SP)

Static pressure is the potential or stored energy of the air often called the “bursting” pressure and this term does in fact make it easier to visualise, because static pressure is the pressure exerted by the air on the walls of the containing vessel. For example when the tyre valve is opened the potential (static) pressure is then converted to kinetic



energy as it flows from the tyre. In the case of a ventilation duct it is the pressure holding the duct open. It is called static pressure because it is the same as the pressure of the air, which would exist if the air were not moving.

In ventilation ducts the static pressure is measured with a **side tube**, normal to the direction of airflow.

3.8.4 Velocity (Dynamic) Pressure (VP)

Moving air possess “kinetic energy” which is the energy associated with motion. In mine ventilation this is termed “Velocity Pressure”. The faster the air moves the greater velocity pressure will be and vice versa. Velocity pressure cannot be measured directly, however it can be measured indirectly using **pitot tube**.

Velocity pressure is calculated from

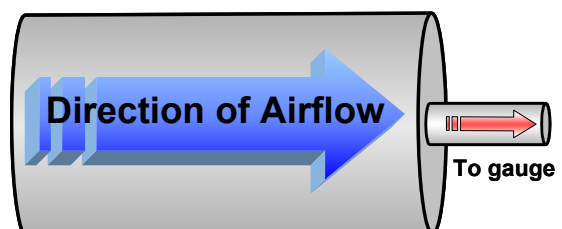
$$VP = \frac{\rho v^2}{2} \quad \text{Equation 3-8 Velocity Pressure}$$

Where

VP	=	Velocity Pressure (Pa)
ρ	=	Density of air (kg/m ³)
v	=	Velocity (m/s)

3.8.5 Total Pressure (TP)

Total Pressure is the algebraic sum of the static pressure (potential energy) and the velocity pressure (kinetic energy) and is measured with a **facing tube**, parallel to the direction of flow.



$$TP = SP + VP \quad \text{Equation 3-9} \quad \text{Ventilating Pressure}$$

Where

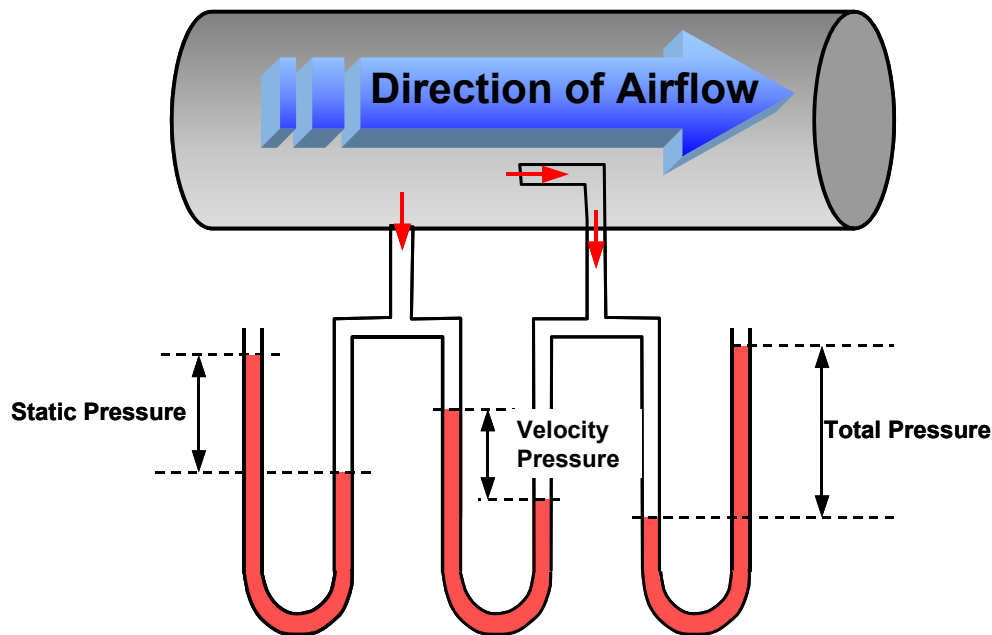
TP = Total Pressure (Pa)

SP = Static Pressure (Pa)

VP = Velocity Pressure (Pa)

3.8.6 Measuring Pressure in a Duct

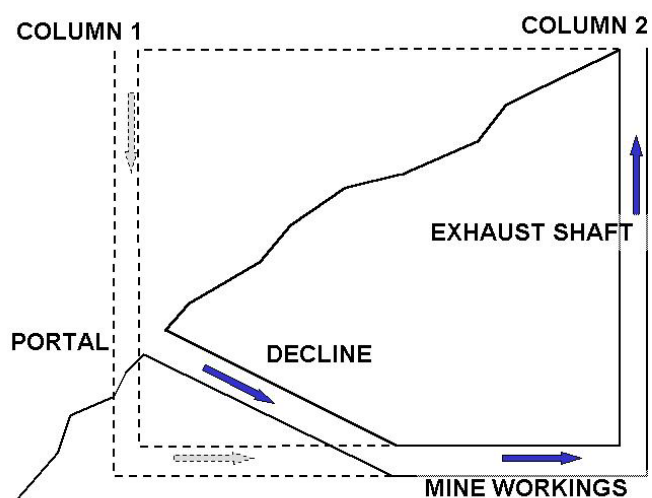
The static, velocity and total pressure may be obtained for airflow in a duct using a tube and a manometer.



3.8.7 Natural Ventilating Pressure

Airflow through a mine will occur whenever there is a pressure difference between the intake and exhaust. Natural Ventilation Pressure (NVP) is formed when there is a difference in air temperature and density between a vertical column of air inside a mine and the corresponding column of air outside the mine. When these columns are connected airflow will occur.

This is often referred to as the chimney effect where warm air rises up the opening displacing the colder air above and drawing more air into the bottom. This method for ventilating underground was



practiced by the Greeks in 600BC when fires were lit at the bottom of shafts to create a draught of air.

In mines this occurs when there is a difference in temperature between the workings and the surface. Variations of temperatures will occur from day to night, summer to winter and may in fact flow in opposite directions. For example the heat of summer may cause the air to flow into the workings, whereas in the colder winter months the flow will be from the mine workings and will have an impact on the operating point of the primary ventilating fans. Although this impact on mine fan performance is limited, the potential NVP needs to be quantified.

Air will flow from the column with the lowest average temperature to the column with the highest average temperature.

The variance of NVP is dependent upon a number of variables including

- elevation
- depth from surface
- geothermal gradient
- local climatic conditions

In its simplest form NVP can be estimated by assuming a closed circuit (intake $\Rightarrow$ workings $\Rightarrow$ exhaust).

$$\text{NVP} = \rho gh \quad \text{Equation 3-10} \quad \text{Natural Ventilating Pressure}$$

Where

NVP = Pascals

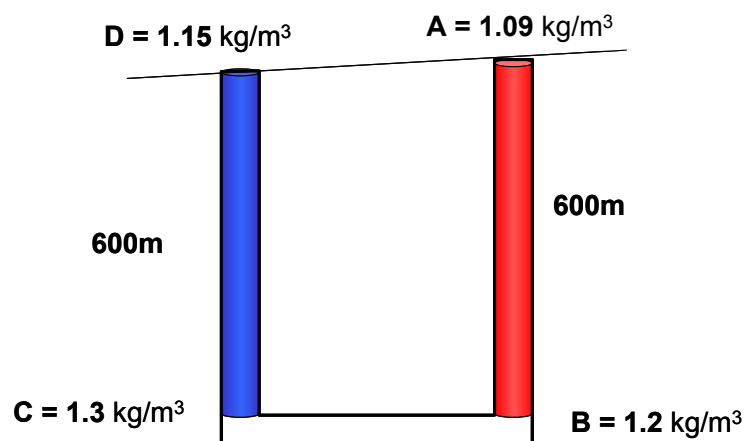
g = Acceleration due to Gravity (9.81m/s^2)

h = Difference in height between start and finish of each branch (m)

ρ = Mean True Density in each branch (kg/m^3)

Example

From the data shown in the schematic, determine the mine Natural Ventilating Pressure in (Pa)



Calculate the mean density in each branch

$$AB = \frac{1.09 + 1.12}{2} = 1.105 \text{ (kg/m}^3\text{)}$$

$$BC = \frac{1.12 + 1.13}{2} = 1.125 \text{ (kg/m}^3\text{)}$$

$$CD = \frac{1.13 + 1.15}{2} = 1.14 \text{ (kg/m}^3\text{)}$$

Calculate NVP in each branch

$$NVP_{AB} = 9.81 \times 600 \times 1.105 = 6504 \text{ (Pa)}$$

$$NVP_{CD} = 9.81 \times 600 \times 1.140 = 6710 \text{ (Pa)}$$

Total Mine NVP

$$\begin{aligned} &= NVP_{AB} - NVP_{BC} - NVP_{CD} \\ &= 6504 - 6710 \\ &= -206 \text{ (Pa)} \end{aligned}$$

For mines with surface intake and exhaust point at the same level, this equation can be simplified to:

$$NVP = gh(\rho_{MI} - \rho_{ME})$$

Where

$$\begin{aligned} \rho_{MI} &= \text{Mean Density of the Intake (kg/m}^3\text{)} \\ \rho_{ME} &= \text{Mean Density of the Exhaust (kg/m}^3\text{)} \end{aligned}$$

3.9 Assignments

3.9.1 Calculate the velocity pressure for air quantities of 10 m³/s, 20 m³/s & 30 m³/s in ducts with diameters of 1,000 mm 1,200 mm & 1,400 mm. Assume the density of air is 1.2 kg/m³

3.9.1 [A] Equations used: 1).... $VP = \frac{\rho v^2}{2}$ (Pa) 2)..... $v = \frac{Q}{A}$ (m/s) 3) $A = \frac{\pi D^2}{4}$ (m²)

Density (kg/m ³)	1.2			
Diameter (m)		1.0	1.2	1.4
Area (m ²)		0.785	1.131	1.539
Q = 10 (m ³ /s)	v (m/s)	12.7	8.8	6.5
	VP (Pa)	97	47	25
Q = 20 (m ³ /s)	v (m/s)	25.5	17.7	13.0
	VP (Pa)	389	188	101
Q = 30 (m ³ /s)	v (m/s)	38.2	26.5	19.5
	VP (Pa)	875	422	228

3.9.2 Using a pitot static tube the following pressure readings were recorded in a flexible duct 1,200 mm diameter.

Measuring Point	1	2	3	4	5	6	7	8
Facing tube	1825	1845	1870	1920	1880	1850	1845	1835
Side tube	1490	1545	1555	1560	1560	1555	1560	1550

If the density of the air was 1.26kg/m³ calculate

1. the average static pressure
2. the average velocity pressure

the quantity of air flowing in the duct

3.9.2 [A] Equations used: 1).... $A = \frac{\pi D^2}{4}$ (m²) 2).... $VP = TP - SP$ 3).... $v = \sqrt{\frac{2VP}{\rho}}$ 4).... $Q = VA$

Area of the duct	1.132 (m ²)							
Measuring Point	1	2	3	4	5	6	7	8
Facing tube	1825	1845	1870	1920	1880	1850	1845	1835
Side tube	1490	1545	1555	1560	1560	1555	1560	1550
VP (Pa)	335	300	315	360	320	295	285	285
v (m/s)	23.1	21.8	22.4	23.9	22.5	21.6	21.3	21.3
Average Static Pressure	312 (Pa)							
Average Velocity	22.2 (m/s)							
Quantity of air	25.1 (m ³ /s)							

“Ventilation is a science, albeit not always a precise science, because in practice there are usually a number of factors involved which cannot be accurately evaluated. A successful mine ventilation engineer uses about fifty percent common sense in solving the problems he is faced with. In addition he uses about forty percent basic knowledge and perhaps ten percent specialised knowledge”

W. L. Le Roux

Mine Ventilation for Beginners 3rd Edition

4 FUNDAMENTALS OF AIRFLOW

The quantity of air that will flow through a “system” (duct, or mine workings) is dependent upon:

- the difference in pressure at the start of the system and the end of the system, and
- the size of the opening.

Again consider the tyre. With the valve opened the air rushes out until the pressure stops (equalises). The more the valve is opened the faster the air escapes even though the pressure inside the tyre was the same to start with. However there are other factors that cause more or less air to flow. These being:

- the roughness of the wall and,
- the severity and number of times the air has to change direction.

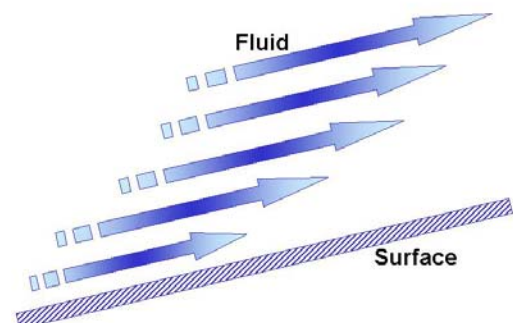
Once something begins to move and the energy remains constant it continues to move at the same velocity. However if the surface roughness changes it becomes difficult to continue at the same velocity and more energy is required to maintain the velocity. The losses in energy due to the roughness of the wall are termed “frictional pressure losses”.

When something is moving in a straight line at a constant velocity, energy is required to change the velocity or direction. These changes occur whenever the airway in which it is flowing changes direction, shape or size. The losses in energy due to a change in direction of the airflow are termed “shock pressure losses”.

In a mine there is a constant conversion of energy from potential (static pressure) to kinetic (velocity pressure) and the energy source must be maintained to ensure that the air continues to flow. If the energy source is removed the flow will stop.

Air as we know contains water (humidity) in various quantities. Although not strictly a fluid, there are some similarities. Lets now consider air as a fluid described as a substance, which deforms continuously when subjected to shear stress.

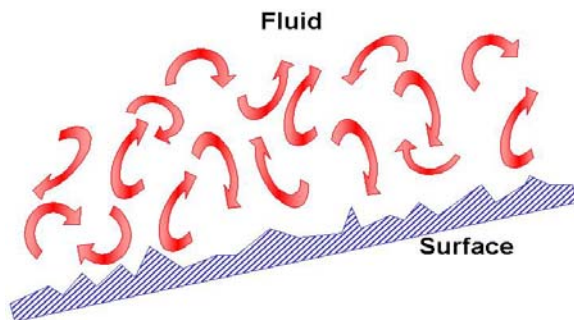
To explain, assume the fluid consists of layers parallel to each other and a force is applied, in a direction parallel to its plane. This force divided by the area of the layer is called shear stress and as long as it is applied, the fluid will flow relative to the other layers.



4.1.1 Laminar Flow

At low fluid velocities the streamlines of flow are almost parallel to each other. The shear resistance between these streamlines is caused by friction between the layers moving at different velocities. If there is no resistance to movement, the fluid is called frictionless or ideal. If the shear movement is resisted then the fluid is called real. In reality ideal fluids do not exist, however in some cases the resistance is small enough to be insignificant.

4.1.2 Turbulent Flow



As the velocity is increased, the streamlines become randomly arranged and the flow becomes turbulent with the additional eddy currents adding to the shear resistance.

4.2 Airflow Equation

Investigations undertaken in the 1800's recognised that if there is no pressure difference between the start and end of an airway, there will be no flow of air; i.e. if the pressure was equal to zero, then the quantity of air flowing was also equal to zero. They also recognised that as the pressure was increased then the airflow quantity also increased i.e. the pressure is proportional to the quantity of airflow. This relationship was termed the resistance of the system and was expressed as

$$R = \frac{P}{Q}$$

Where

R	=	Resistance (Ns ² /m ⁸)
P	=	Pressure (Pa)
Q	=	Quantity of airflow (m ³)

This held true whilst the airflow was laminar however, it did not hold true for turbulent airflow and it was noted that in fully turbulent flows that to double the quantity of air flowing then a pressure four times the original was required. Similarly, to increase the quantity three-fold, the pressure required was nine times the original pressure. In other words, the pressure required increases as the square of the quantity¹¹ (i.e. Pressure is proportional to the quantity squared). This relationship has since been called the Ventilation Equation.

$$P = RQ^2 \quad \text{Equation 11 Ventilation Equation}$$

There can be speculation that, when air flows at low velocity through a worked-out area, or where low velocity leakage occurs, the flow pattern is streamlined (laminar) and therefore the

¹¹ Numerous investigations showed that the index actually lies in the range 1.8 to 2.2 for underground mines and approaching 1 in large voids where the airflow becomes laminar. In the vast majority of cases the use of 2 is accepted as the standard.

formula relating P, R and Q should revert to $P = RQ$. Although this is true, in reality the pressure differentials are small enough to be inconsequential and it is generally accepted in mine ventilation that all flows are turbulent and this relationship is seldom, if ever, used.

Example

Calculate the pressure loss when 4 m³/s of air flows through a duct having a resistance of 9.3 Ns²/m⁸

From the ventilation equation

$$\begin{aligned} P &= R Q^2 \\ &= 9.3 \times 4^2 \\ &= 149 \text{ Pa} \end{aligned}$$

Example

Calculate the pressure loss when 8 m³/s of air flows through the same duct.

From the ventilation equation

$$\begin{aligned} P &= R Q^2 \\ &= 9.3 \times 8^2 \\ &= 595 \text{ Pa} \end{aligned}$$

You should note that in the second example the quantity flowing is double that flowing in the first example and that the pressure required for this increased flow is four times that required in the first case.

4.3 Resistance (The friction factor).

In around 1850 Atkinson hypothesised that the value of R was dependant upon on certain characteristics of the airway or duct. For example, if one airway has a small cross sectional area and another a large cross sectional area, and all other factors (P and Q) remain constant, air will flow more easily through the second airway. In other words, the larger the cross sectional area (A) of the airway, the lower the resistance (R) of the airway.

Most of the early work on fluid flow was undertaken on the assumption of circular pipes and showed that the drag force (resistance = R) of a pipe depends on the flow velocity (v), the density of the fluid (ρ) and the cross sectional area (A) of the pipe. We can therefore say that $R = f(v, \rho, A)$ where f is some function of the variables. By substituting C_1 for f and completing a dimensional analysis the equation for this relationship can be written as $R = C_1 v^2 \rho A$ and would assume C_1 as a constant. Measurements on different pipes and various velocities would show that C_1 is NOT constant and there was a fourth variable, i.e. the viscosity (μ) of the fluid and the relationship for R is rewritten as $R = f(v, \rho, A, \mu)$. Since there are four unknowns it is not possible to obtain a unique solution. By analysis¹² and substitution the equation is written as

$$\frac{R}{\rho D^2 v^2} = f_1 \left(\frac{\mu}{\rho D v} \right) \text{ or } \frac{R}{\rho D^2 v^2} = \left(\frac{\rho D v}{\mu} \right). \text{ The non-dimensional group } \left(\frac{R}{\rho D^2 v^2} \right) \text{ is referred to as}$$

the force or Euler coefficient, denoted by C_f and the group $\left(\frac{\rho D v}{\mu} \right)$ is referred to as Reynolds Number (R_e).

¹² The dimensional analysis and solution is described in "Environmental Engineering in South African Mines" Chapter 1. (pp1-27)

However in addition to Reynolds Number the surface roughness also influences the resistance force. The surface roughness (e) also has a relationship to the area or as above to the diameter (D), and this is described as the 'relative roughness' and defined as $\varepsilon = \frac{e}{D}$.

Further evaluation and relating to Bernoulli's Equation¹³ $\left[P_1 + \rho \frac{v_1^2}{2} + \rho g H_1 = P_2 + \rho \frac{v_2^2}{2} + \rho g H_2 \right]$

it is shown that $k = \frac{P}{\frac{1}{2}\rho v^2}$ and k is described as the friction component and expressed as

$\lambda = \frac{P}{\frac{1}{2}\rho v^2 \frac{L}{D}}$ where P is the static pressure loss due to wall friction in a parallel duct diameter D

and length L .

It is also possible to equate this to rectangular ducts if the diameter (D) is replaced with the equivalent hydraulic diameter D_h

$$D_h = \frac{4A}{C} \quad \text{Equation 12 Hydraulic Diameter}$$

Where D_h = Hydraulic diameter (m)
 A = Cross sectional Area of Airway (m^2)
 C = Circumference of Airway (m)

The equation for friction can now be rewritten as $\lambda = \frac{P}{\left(\frac{\rho v^2}{2} \left(\frac{L}{D_h} \right) \right)}$

A more complete analysis will show that the friction coefficient is a function of Reynolds Number (R_e), relative roughness (ε) and duct shape (s) where

$$\text{Reynolds Number } = R_e = \left(\frac{\rho D v}{\mu} \right)$$

Surface roughness = $\varepsilon = \frac{e}{D}$ and

s = the shape factor of the duct cross section.

Experiments showed that the shape factor (s) had very little influence and the non-dimensional parameters had a major influence. After further studies and experimental work on smooth and rough walled ducts.

The equation for the actual relationship shown in this test work can be represented by

$$\frac{1}{\sqrt{\lambda}} = 1.74 - 2 \log \left(2\varepsilon + \frac{18.7}{Re \sqrt{\lambda}} \right) \quad \text{Equation 13 Friction Factor}$$

Where

¹³ Bernoulli's Equation : Law of Conservation of Momentum is not discussed in these notes but can be found in many texts dealing with fluid flow.

$$\varepsilon = \frac{e}{D_h} = \text{Relative roughness}$$

e = Average height of surface irregularities or surface roughness
(assume 0.35 m for rock wall, 0.02 m for raisebored)

D_h = Hydraulic Diameter (m)

$$\text{And } Re = \frac{\rho D_v}{\mu}$$

ρ = Density (kg/m³)

μ = Viscosity (kg/ms)

D_h = Hydraulic Diameter (m)

V = Velocity (m/s)

In mine ventilation the term λ is usually replaced by 'k' known as the coefficient of friction or Atkinson's k Factor.

4.3.1 Atkinson's Equation

Atkinson also noted that if air has to rub against a larger area of surface in one airway than in another, the resistance will be higher in the airway with the largest "rubbing surface". This rubbing surface is found by multiplying the perimeter (C) by the length (L).

$$P = \frac{kCLv^2}{A} \quad \text{Equation 14 Atkinson's Ventilation Equation (Velocity)}$$

Where

P	=	Pressure (Pa)
k	=	Atkinson's Friction Factor with consideration to surface roughness.
C	=	Circumference of the duct (m)
L	=	Length of duct m)
A	=	Cross sectional Area of the duct (m ²)
v	=	Velocity of the fluid (m/s)

If we substitute the physical values in the Atkinson equation

$$\text{Left hand side Pressure is } Pa = \frac{N}{m^2} \text{ and the Right hand side } \frac{kCLv^2}{A} = \frac{k \times m \times m \times \frac{m^2}{s^2}}{m^2} = \frac{km^2}{s^2}.$$

$$\text{Combining both sides } \frac{N}{m^2} = \frac{km^2}{s^2}$$

$$\text{therefore } k = \frac{Ns^2}{m^4}$$

$$\text{Substituting the dimensions of force } N = \frac{kg \times m}{s^2} \text{ then } k = \left(\frac{kg \times m}{s^2} \right) \times \left(\frac{s^2}{m^4} \right) = \frac{kg}{m^3}$$

This is important because it shows that Atkinson's friction factor is not dimensionless and will vary according to the dimensions $\frac{kg}{m^3}$ which is the same dimensions as density. Atkinson had

ignored density effects deriving this equation and therefore is valid only for air with standard density.

The use Atkinson's equation for any density it need to have a correction applied and it becomes

$$P = \frac{kCLv^2}{A} \times \frac{\rho}{\rho_s}$$

Where ρ = Density if air flowing in the duct (kg/m^3)
 ρ_s = Density of standard air (1.2 kg/m^3)

The term $\frac{\rho}{\rho_s}$ is in Atkinson's formula to allow for the fact that the pressure requirements depend on the density of the air. Obviously, more pressure will be required to pass heavier (denser) air through a system than the lighter air. In fact, the pressure (P) required is directly proportional to the air density (ρ).

More often than not it is convenient to express Atkinson's equation in terms of quantity of air rather than velocity and since $V = QA$ this equation is rewritten as;

$$P = \frac{kCLQ^2}{A^3} \times \frac{\rho}{\rho_s} \quad \text{Equation 15 Atkinson's Ventilation Equation (Quantity)}$$

Where

P = Pressure (Pa)
 K = Atkinson's Friction Factor (Ns^2/m^4)
 C = Circumference of Airway (m)
 L = Length of Airway (m)
 A = Cross sectional Area of Airway (m^2)
 Q = Quantity of airflow (m^3/s)
 ρ = Density if air flowing in the duct (kg/m^3)
 ρ_s = Density of standard air (1.2 kg/m^3)

The values of 'k' are determined from measurement and calculation. The values below are sighted in texts, and generally hold up to scrutiny.

Some mines may find differing results outside of these values. In practice all mine should measure and determine their specific factor(s) because there will always be more than one factor that could be used.

Some Typical Values for Atkinson's Friction Factor (k) at Standard air density

Airway Type.	k factor (Ns^2/m^4)
Smooth pipe	0.0028
Normal rigid ducting	0.0030 to 0.0035
Flexible ducting	0.0030 to 0.0065
Concrete surfaced	0.0035 to 0.0040
Rock surfaced	0.0070 to 0.0200
Raise bored	0.0035 to 0.0050
Timbered Airway	0.0400 to 0.0600
Timbered rectangular Shaft	0.0400 to 0.1000

Example

Calculate the pressure required to overcome friction with an airflow of 60 m³/s in a 5.0 m x 4.0 m haulage drive that is 450 m long and has a friction factor (k) of 0.016 Ns²/m⁴. Air density in the haulage is 1.1 kg/m³.

Given that

$$\begin{aligned} k &= 0.016 \text{ Ns}^2/\text{m}^4 \\ L &= 450 \text{ m} \\ \rho &= 1.1 \text{ kg/m}^3 \\ Q &= 60 \text{ m}^3/\text{s} \\ C &= 2(5 + 4) = 18 \text{ m} \\ A &= (5.0 \times 4.0) = 20.0 \text{ m}^2 \end{aligned}$$

$$\begin{aligned} \text{From Atkinson's equation } P &= \frac{kCLQ^2}{A^3} \times \frac{\rho}{\rho_s} \\ &= \frac{0.16 \times 18.0 \times 450 \times (60)^2}{20.0^3} \times \frac{1.1}{1.2} \\ &= 53.5 \text{ Pa} \end{aligned}$$

Therefore, the pressure required to overcome the friction in this drive is 54 Pa

4.4 Shock Losses

Shock losses are the change in total pressure across all airway elements such as the entrance to a system, a bend, junction, obstruction, change in section and exit from the system. In short shock losses are the result of flow separation that occurs whenever the airflow changes direction.

The pressure losses resulting from a change in direction can be determined from the **Shock Loss Equation**

$$P_{\text{SHOCK}} = X \times VP$$

Where

P_{SHOCK} = Pressure loss due to shock (Pa)

X = Dimensionless shock loss factor similar to Atkinson's friction factor and is constant only for a given set of conditions depending upon shape, dimensions and characteristics. (See below)

VP = Velocity Pressure of the air (Pa)

And
$$VP = \rho \times \frac{v^2}{2}$$

Where

ρ = Density of the air (kg/m³)

v = Velocity of the air (m/s)

The shock loss factor X is a function of the:

1. Configuration and flow through the element.
2. Angle of the change in direction,

3. Degree the abruptness of the change,
4. Radius of curvature,
5. Ratio of the radius to width of the airway
6. Aspect ratio between the height and width,
7. Velocity in the airway
8. Airway roughness
9. Shape of the airway
10. Airways immediately before and immediately after the change in direction,
11. The number and type of complex elements i.e. two bends, bend followed by an expansion

The large number of variables that contribute to shock losses causes the calculation of these losses is extremely complicated and seldom necessary in mine ventilation because of their small magnitude and the inaccuracy of the X value.

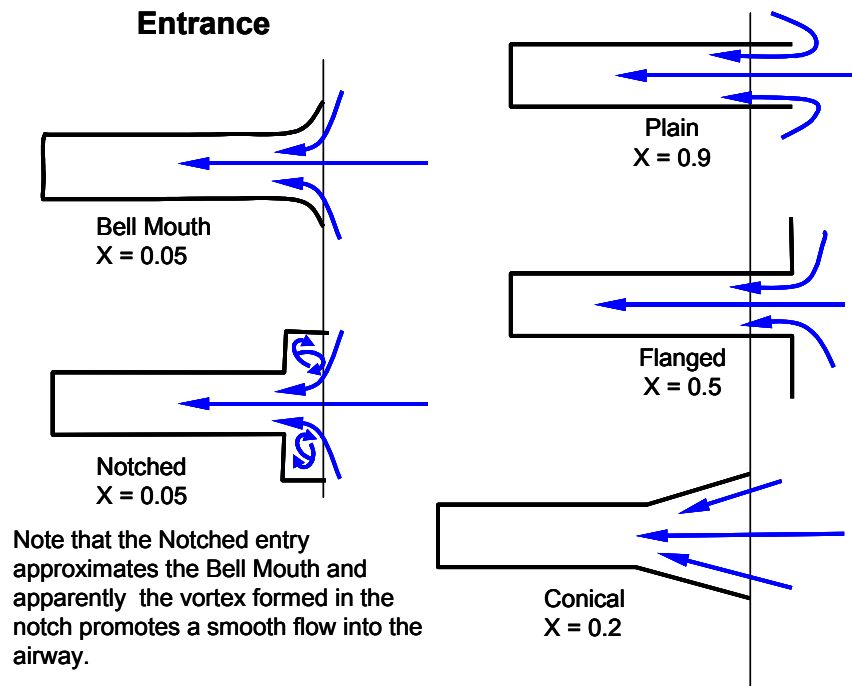
WARNING- If airflow and pressure surveys have been used to determine the resistance of any part of the mine workings these measurements will include the losses due to shock and any use of additional factors will cause undetectable errors in the results.

Shock losses are generally “over used” in computer ventilation modelling and the calculation methods shown below are approximations of the actual number taken from published texts¹⁴ and suitable for use in most computer ventilation models. Wherever possible ventilation practitioners should determine these values by direct measurement.

¹⁴“Fan Engineering – Eight Edition” Published by Buffalo Forge Company. Buffalo, New York USA (1983) & DALY, B.B. “Woods Practical Guide to Fan Engineering” Published by Woods of Colchester Limited UK (1978)

4.4.1 Entrance

Shock losses occur at the entrance to any system due to the acceleration of air from zero velocity in the surrounding atmosphere to the velocity in the airway.



4.4.2 Outlet Losses

Air exiting a system decelerates from the airway velocity to zero velocity in the atmosphere.

The static pressure in the issuing air stream will be equal to the surrounding atmospheric pressure. If this is taken as a zero gauge pressure then the total pressure at the exit will be equal to the velocity pressure.

To reduce this loss, the area of the outlet must be increased (ie the velocity is decreased) this is normally achieved with the use of a diffuser or evasè.

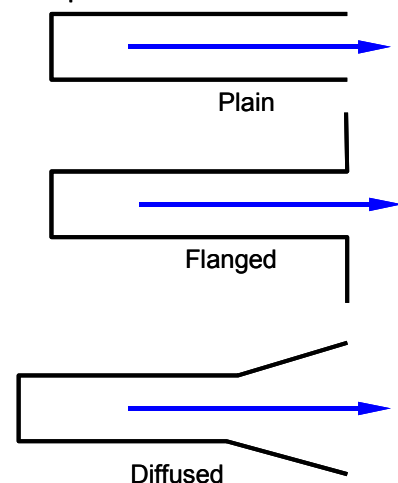
The performance of a diffuser will vary with the inlet velocity pressure and the ratio of the outlet to inlet area, the centreline length and the angle of divergence. In an ideal diffuser the regain pressure would be equal to the change in velocity pressure and the loss in total pressure (shock loss) would be zero. In reality this will never occur.

A well-designed diffuser would have an outlet to inlet area ratio of 4:1 with an included angle of divergence between 8° and 11° . In practice the length required to regain pressure is excessive and generally the divergence angle is around 25° .

As a rule of thumb if the length of the divergence is four times the diameter of the smaller airway, then the shock losses become negligible because the pressure regain of shock loss pressures are overcome by the frictional pressure losses.

Outlets

In all cases $X = 1.0$ and the velocity pressure is determined by the velocity in the plane of the exit area



4.4.3 Elbows

Elbows, bends and mitres are used to guide the change in direction of flow and restrict flow separation. The relative losses will depend upon the abruptness of the change in direction, Reynolds number (not discussed in these notes) and the wall roughness of the airway. Other considerations include geometrical shape i.e. circular or round, the angle of the bend and the radius ratio and the aspect ratio.

Those of most importance in mines are mitre bends with curved bends having a lesser impact due primarily to their lengths such that shock losses are decreased to a point of insignificance compared to the frictional pressure losses.

Consider the case of a typical mine curve of radius 20 metres with a height of 4.5 metres and a width of 4.5 metres with a friction factor “k” of $0.015 \text{ N s}^2 \text{ m}^{-4}$. The angle of the curve is 90° and the airflow is $200 \text{ m}^3/\text{s}$

The calculation for “X” in a smooth bend is described by McElroy (1935)

$$X = \frac{0.60}{\text{m}^2 \times a^{0.5}} \times \left(\frac{\theta}{90} \right)^2$$

Where

$$\begin{aligned} \text{m is the radius ratio} &= \frac{\text{Centre line radius}}{\text{Width of the Airway}} \\ &= \frac{4.5}{2} \\ &= 2.25 \end{aligned}$$

$$\begin{aligned} \text{a is the aspect ratio} &= \frac{\text{Height of the Airway}}{\text{Width of the Airway}} \\ &= \frac{4.5}{4.5} \\ &= 1.0 \end{aligned}$$

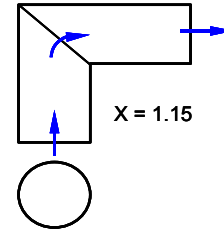
$$\begin{aligned} \text{Therefore } X &= \frac{0.6}{2.25^2 \times 1.0^{0.5}} \times \left(\frac{90}{90} \right)^2 \\ &= 1.2 \end{aligned}$$

The velocity is $200/(4.5 \times 4.5) = 9.87 \text{ (m/s)}$

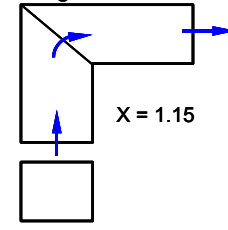
Therefore

$$\begin{aligned} P_{\text{SHOCK}} &= 1.2 \times (0.5 \times 1.2 \times 9.87^2) \\ &= 70 \text{ Pa} \end{aligned}$$

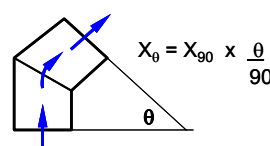
Circular Mitre Bend



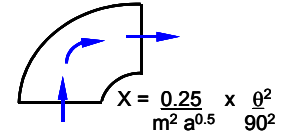
Rectangular Mitre Bend



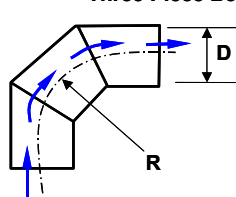
Mitre Bend other than 90°



Smooth Bend

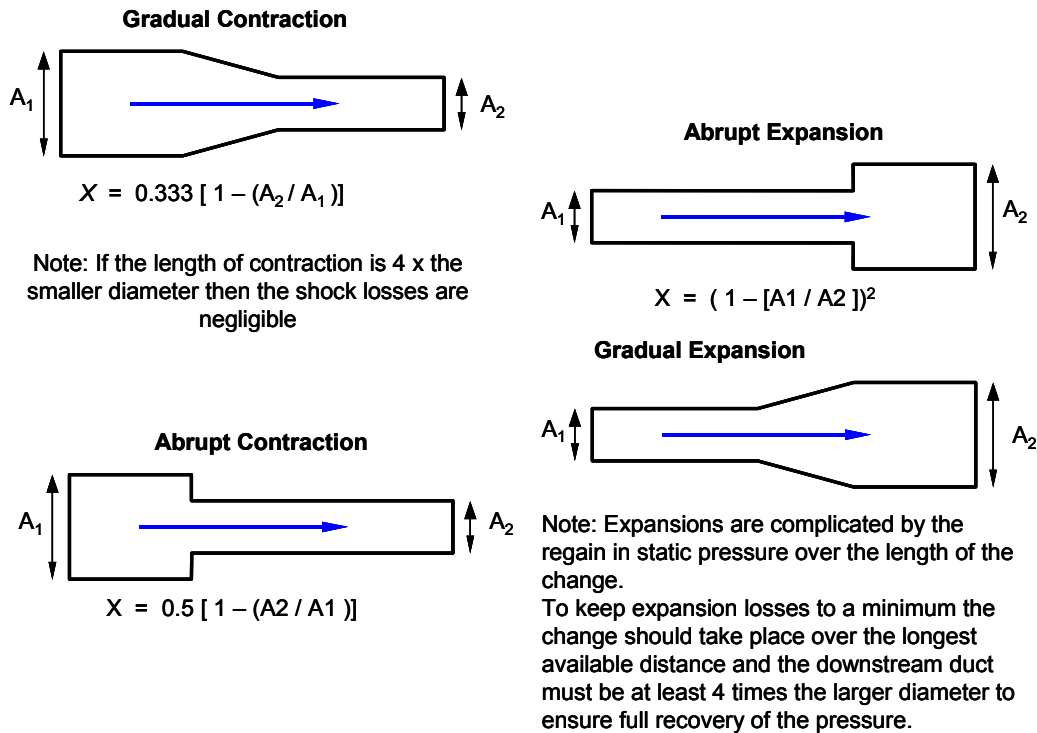


Three Piece Bend 90°



R/D	0.5	1.0	1.5	2.0	2.5	3.0
X	0.425	0.350	0.315	0.315	0.325	0.380

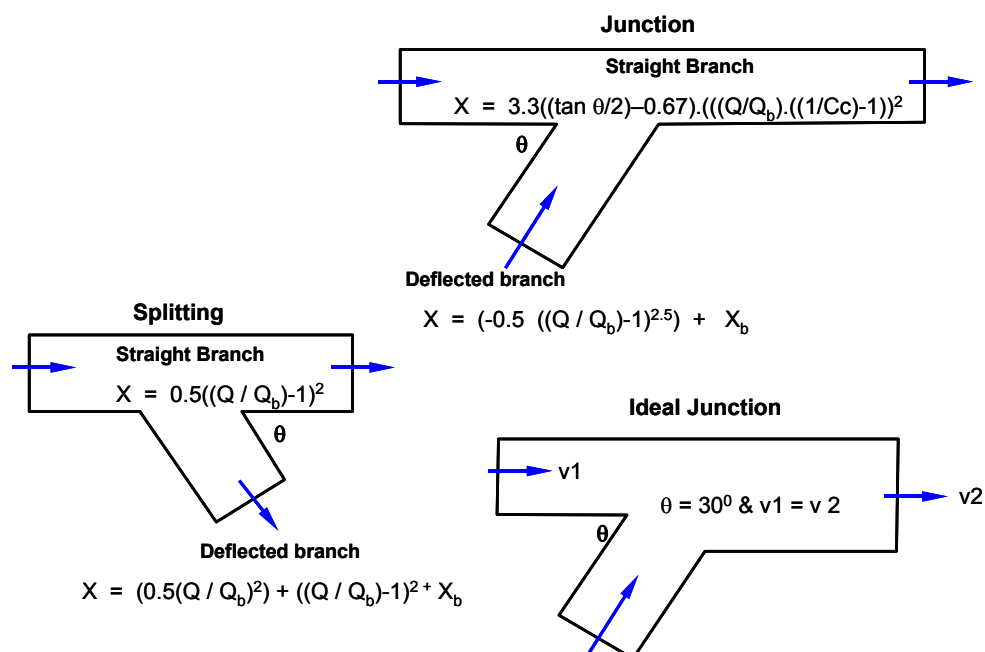
4.4.4 Expansions and Contractions



4.4.5 Junctions and Splitting

These losses depend upon

1. the geometrical aspect of each airway,
2. the branching angle,
3. the branch area ratio,
4. the ratio of the downstream to upstream velocity of air and therefore the quantity of airflow.



In the deflected branch of a split the pressure loss can be reduced considerably by using a converging taper between the main and the deflected branch.

The recommended procedure for development of a junction is to join the upstream main branch with the down stream main by means of a tapered convergence of at least two upstream diameters long made to join the branch to the taper at an angle of 30° with the upstream main branch. The idea is to maintain a constant velocity in the main branch. Obviously this is fine for a duct system in a building but extremely difficult to achieve economically in an underground mine.

One important aspect of junctions is that the pressure loss may in fact be negative depending of the ratio of the branch flow to the total flow. This is due to the transfer of momentum from one branch to the other.

In the equations for splits and junctions the following applies:

Q = Total Airflow

Q_b = Airflow in the branch being evaluated

X_b = X for the deflected branch

C_c = Coefficient of Contraction

$$= \frac{A_c}{A_o}$$

And A_c = Area of vena contractor

A_o = Area of orifice

4.5 Other Methods of Expressing the Shock Factor X

Some attempt to **increase the friction** “ k ” as a method to allow for shock losses. This involves the addition of an increment to the friction factor “ k ” for each and every airway. The method is tedious and prone to error due to the many variables and is not really recommended for use.

This method is inadvertently applied when the results of pressure quantity surveys are used to “back calculate” the friction factor “ k ” because the pressure losses due to shock are included in the measurements.

Use of the **equivalent-length method** is probably the most useful method for mine ventilation because it can be calculated by equating the formulas for friction loss and shock loss.

To differentiate between the actual length (L) the subscript e is added to indicate equivalent length $L_e = \frac{X \times D_h}{6.67k}$ where D_h is the hydraulic diameter of the airway. $D_h = \frac{4A}{C}$.

WARNING - Mine ventilation computer programmes may include preset values for shock losses. These values are expressed as an equivalent length (L_e) of the airway. It is then used to calculate a resistance for the L_e and is then added to the resistance of the airway. Some of these values have been taken from a text that used drive sizes of only 2.0m x 2.0m and therefore are not applicable to all mines.

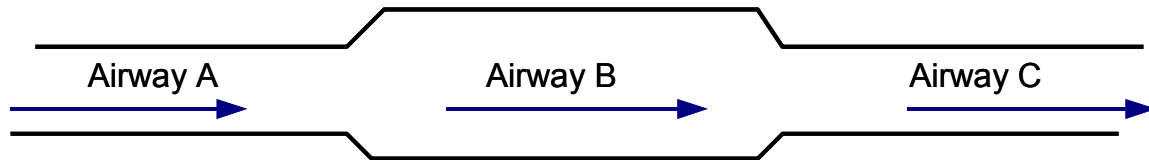
“Shock losses do not lend themselves to precise calculation because of the great range of variability in occurrence and because of a lack of understanding of their very nature.”¹⁵

¹⁵ HARTMAN, H.L. “Mine Ventilation and Air-conditioning”

4.6 Series Circuits

When one airway is followed by another airway and both have their own physical characteristics, the airways are said to be in series.

This becomes obvious when we refer to the sketch below where the total quantity of air flowing through the system is equal to the quantity of air in 'A', which is equal to the quantity of air flowing in 'B' and likewise in airway 'C'. i.e. $Q_T = Q_A = Q_B = Q_C$



The total pressure loss in the system is the sum of the pressure losses in airway 'A', airway 'B' and airway 'C'. i.e. $P_T = P_A + P_B + P_C$

It then follows that the total resistance of the system is the sum of the resistance of airway 'A' and airway 'B' and airway 'C'. i.e. $R_T = R_A + R_B + R_C$

This relationship is usually expressed as

$$R_T = R_1 + R_2 + \dots + R_n$$

Where $R = \text{Resistance (Ns}^2/\text{m}^8\text{)}$

Example

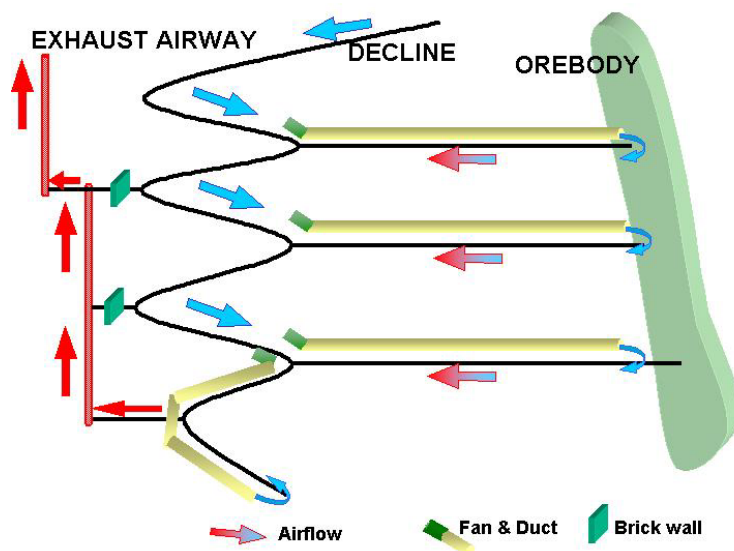
Referring to the sketch above Airway A has a resistance of $0.10 \text{ (Ns}^2/\text{m}^8\text{)}$, Airway B a resistance of $0.07 \text{ (Ns}^2/\text{m}^8\text{)}$ and Airway C a resistance of $0.17 \text{ (Ns}^2/\text{m}^8\text{)}$. Calculate the pressure (P) required to cause a flow of $100 \text{ m}^3/\text{s}$

$$\begin{aligned} R_T &= R_1 + R_2 \\ &= 0.10 + 0.07 + 0.17 \\ &= 0.34 \text{ (Ns}^2/\text{m}^8\text{)} \end{aligned}$$

and

$$\begin{aligned} P_T &= R_T \times Q_T^2 \\ &= 0.34 \times (100)^2 \\ &= 3400 \text{ (Pa)} \end{aligned}$$

Many Australian mines utilise “**series ventilation**” circuits in their design. This is a term used to describe a combination of airways with different resistances where the air flows through each in turn. This is a common practice in steeply dipping narrow vein mines, where the air flows from the surface to the lowest working

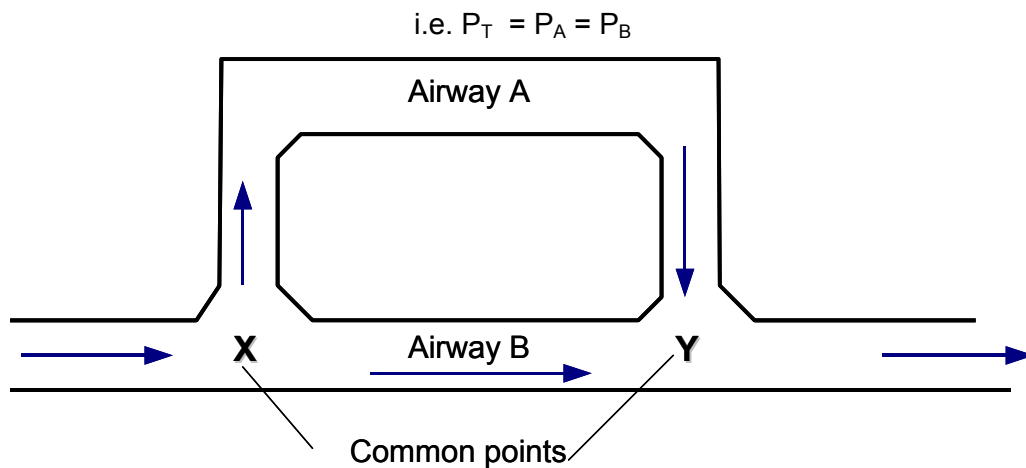


level before being exhausted from the mine. This “reuse” of air is often confused when people talk of recirculation of air when in fact they mean re-used (series) air. Recirculate implies that the air goes around in circles and never leaves the circuit.

4.7 Parallel Airway Circuits

Airways are in parallel when they have a **common inlet** and **common outlet**.

In the sketch to the right the points **X** and **Y** are common to both airways. Therefore the pressure loss across airway A is equal to the pressure loss across airway B and equal to the pressure loss across the system X-Y

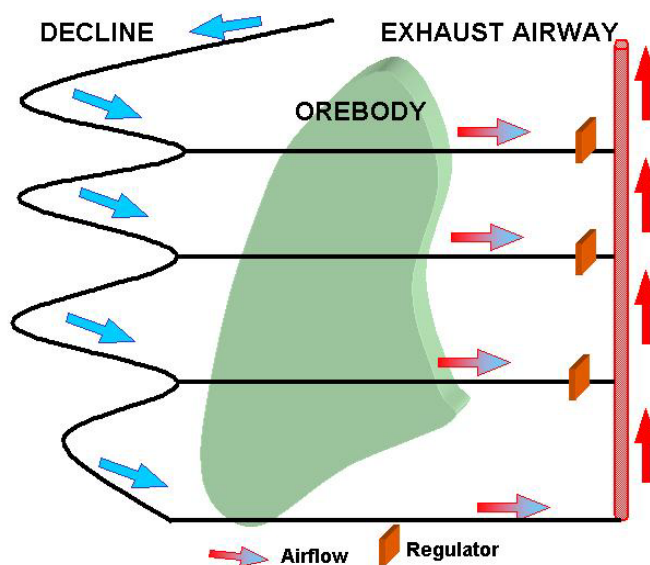


The total quantity flowing in the system is equal to the quantity of air flowing in airway A and airway B. i.e. $Q_T = Q_A + Q_B$ and the overall resistance (R_T) is given by the formula

$$\frac{1}{\sqrt{R_T}} = \frac{1}{\sqrt{R_1}} + \frac{1}{\sqrt{R_2}} + \dots \frac{1}{\sqrt{R_n}}$$

Where R = Resistance (Ns^2/m^8)

Parallel systems exist in mines when air is exhausted on a number of levels.



EXAMPLE

Three identical airways are developed in parallel. Each airway has a resistance of 0.15 (Ns^2/m^8). Calculate the pressure required to cause a flow of 50 m^3/s through each airway.

Calculate the total airflow

$$\begin{aligned} Q_T &= Q_A + Q_B + Q_C \\ Q_T &= 50 + 50 + 50 \\ &= 150 \text{ m}^3/\text{s}. \end{aligned}$$

Calculate the overall resistance (R_T)

$$\begin{aligned} \frac{1}{\sqrt{R_T}} &= \frac{1}{\sqrt{R_A}} + \frac{1}{\sqrt{R_B}} + \frac{1}{\sqrt{R_C}} \\ \frac{1}{\sqrt{R_T}} &= \frac{1}{\sqrt{0.15}} + \frac{1}{\sqrt{0.15}} + \frac{1}{\sqrt{0.15}} \\ &= 0.0167 \text{ (Ns}^2/\text{m}^4) \end{aligned}$$

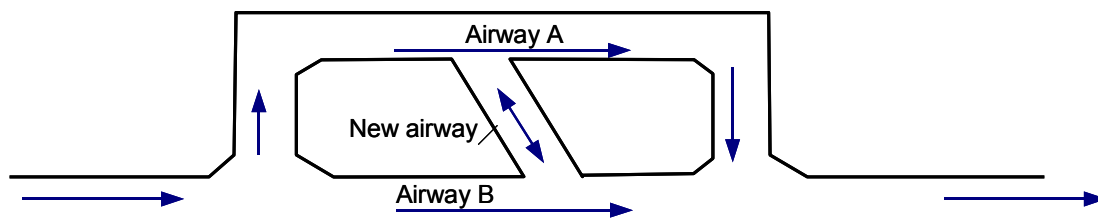
Calculate the pressure

$$\begin{aligned} P_T &= R_T \times Q_T^2 \\ &= 0.0167 \times 150^2 \\ &= 375 \text{ Pa} \end{aligned}$$

4.8 Complex Circuit (Networks)

With reference to the sketch for parallel airways it is noted that whilst Airway A and Airway B are in parallel they are also in series with the airway preceding point X and the airway following point Y.

Lets now consider the same sketch a new airway that connects Airway A with Airway B.



After examining this sketch you will come to the conclusion It is not possible to reduce this circuit to a series or parallel circuit and an analytical or reiterative technique is required.

Complex circuits or networks rely on the use of computers to undertake the numerous calculations required.

In order to understand the complexity of network problems the following definitions apply

1. A network is any complex ventilation circuit.
2. A Junction is where two or more airways join
3. A branch is any airway between two junctions
4. A mesh is any closed path that traverses the network. Every airway must be included once in any group of meshes.

5. Kirchoff's electrical laws can be restated and will apply.

Kirchoff's 1st Law

The algebraic sum of all air quantities at a junction is zero

Kirchoff's 2nd Law

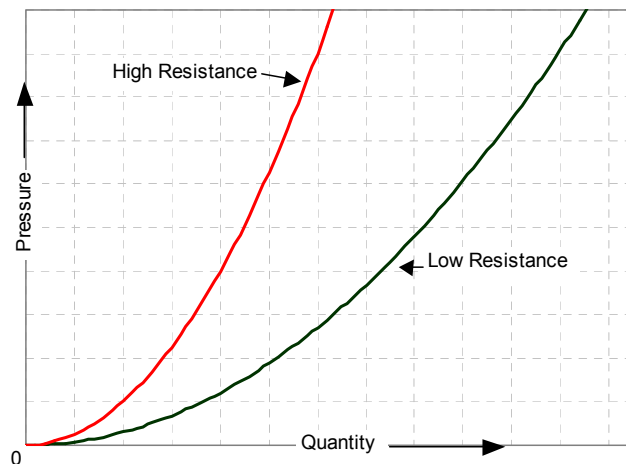
The algebraic sum of all pressure losses in a closed mesh is zero. However if a fan is included in the mesh then the algebraic sum of all pressure losses is equal to the pressure provided by the fan.

Comprehensive discussion of Network analysis can be found in Chapter 8 of "Environmental Engineering in South African Mines".

4.9 System Resistance Curve

We have seen and discussed the relationship of pressure (P), quantity (Q) and resistance (R) in the previous sections. For example the ventilation equation ($P = RQ^2$) shows the relationship between pressure (P) and quantity (Q) is proportional i.e. $\frac{P}{Q^2} = R$ where R was the resistance for a particular system with a specific pressure and quantity.

The "system") resistance in any duct or mine, is determined by summation of all the appropriate resistance values. Once this calculation is complete we can determine the pressure necessary to cause various quantities of air to flow. . The results of these calculations are then shown graphically to produce the "System Resistance Characteristic". A system with a high resistance will be represented by a curve that is steeper than a system with a lower resistance.



4.10 Assignments

- 4.10.1 [Q] Calculate the pressure loss in 250 m of a rigid plastic duct with diameter of 1,200 mm & 1,400 mm for air quantities of 15 m³/s & 25 m³/s. State the k factors used and assume the density of air is 1.2 kg/m³.

- 4.10.1 [A] Equations used: 1) $P = \frac{kCLQ^2}{A^3} \times \frac{\rho}{\rho_s}$ 2) $A = \frac{\pi D^2}{4} (\text{m}^2)$ 3) $C = \pi D (\text{m})$

Determine area (A) and circumference (C) for ducts

Diameter (mm)	C (m)	A (m ²)
1,200	3.7699	1.1310
1,400	4.3982	1.5394

Calculate pressure assuming k = 0.0035

$$P = \frac{kCLQ^2}{A^3} \times \frac{\rho}{\rho_s}$$

1,200 Diameter $P = \frac{0.0035 \times 3.7699 \times 250 \times 15^2}{1.1310^3} \times \frac{1.2}{1.2} = 513 \text{ Pa}$

1,200 Diameter $P = \frac{0.0035 \times 3.7699 \times 250 \times 25^2}{1.1310^3} \times \frac{1.2}{1.2} = 1425 \text{ Pa}$

1,400 Diameter $P = \frac{0.0035 \times 4.3982 \times 250 \times 15^2}{1.5394^3} \times \frac{1.2}{1.2} = 237 \text{ Pa}$

1,400 Diameter $P = \frac{0.0035 \times 4.3982 \times 250 \times 25^2}{1.5394^3} \times \frac{1.2}{1.2} = 659 \text{ Pa}$

- 4.10.2 [Q] Repeat 4.10.1 but assume an air density of 1.15 kg/m³ and 1.25 kg/m³

4.10.2 [A] 1,200 Diameter $P = \frac{0.0035 \times 3.7699 \times 250 \times 15^2}{1.1310^3} \times \frac{1.15}{1.2} = 492 \text{ Pa}$

1,200 Diameter $P = \frac{0.0035 \times 3.7699 \times 250 \times 25^2}{1.1310^3} \times \frac{1.15}{1.2} = 1365 \text{ Pa}$

1,400 Diameter $P = \frac{0.0035 \times 4.3982 \times 250 \times 15^2}{1.5394^3} \times \frac{1.15}{1.2} = 227 \text{ Pa}$

1,400 Diameter $P = \frac{0.0035 \times 4.3982 \times 250 \times 25^2}{1.5394^3} \times \frac{1.15}{1.2} = 632 \text{ Pa}$

1,200 Diameter $P = \frac{0.0035 \times 3.7699 \times 250 \times 15^2}{1.1310^3} \times \frac{1.25}{1.2} = 534 \text{ Pa}$

$$1,200 \text{ Diameter} \quad P = \frac{0.0035 \times 3.7699 \times 250 \times 25^2}{1.1310^3} \times \frac{1.25}{1.2} = 1484 \text{ Pa}$$

$$1,400 \text{ Diameter} \quad P = \frac{0.0035 \times 4.3982 \times 250 \times 15^2}{1.5394^3} \times \frac{1.25}{1.2} = 247 \text{ Pa}$$

$$1,400 \text{ Diameter} \quad P = \frac{0.0035 \times 4.3982 \times 250 \times 25^2}{1.5394^3} \times \frac{1.25}{1.2} = 686 \text{ Pa}$$

4.10.3 [Q] Repeat 4.10.1 using flexible ducting.

4.10.3 [A] Calculate pressure assuming $k = 0.0065$

$$P = \frac{kCLQ^2}{A^3} \times \frac{\rho}{\rho_s}$$

$$1,200 \text{ Diameter} \quad P = \frac{0.0065 \times 3.7699 \times 250 \times 15^2}{1.1310^3} \times \frac{1.2}{1.2} = 953 \text{ Pa}$$

$$1,200 \text{ Diameter} \quad P = \frac{0.0065 \times 3.7699 \times 250 \times 25^2}{1.1310^3} \times \frac{1.2}{1.2} = 2647 \text{ Pa}$$

$$1,400 \text{ Diameter} \quad P = \frac{0.0065 \times 4.3982 \times 250 \times 15^2}{1.5394^3} \times \frac{1.2}{1.2} = 441 \text{ Pa}$$

$$1,400 \text{ Diameter} \quad P = \frac{0.0065 \times 4.3982 \times 250 \times 25^2}{1.5394^3} \times \frac{1.2}{1.2} = 1225 \text{ Pa}$$

4.10.4 [Q] Calculate the pressure loss for air quantities of $13.5 \text{ m}^3/\text{s}$, $55 \text{ m}^3/\text{s}$ & $165 \text{ m}^3/\text{s}$ in a decline 350 m long 5.5m high and 5.0m wide. State the k factor used and assume the density of air is 1.2 kg/m^3 .

4.10.4 [A] Determine area (A) and circumference (C) for drive

Height (m)	Width (m)	C (m)	A (m^2)
5.5	5.0	21.0	27.5

Calculate pressure assuming $k = 0.012$

$$P = \frac{kCLQ^2}{A^3} \times \frac{\rho}{\rho_s}$$

$$P = \frac{0.012 \times 21.0 \times 350 \times 13.5^2}{27.5^3} \times \frac{1.2}{1.2} = 0.7 \text{ Pa}$$

$$P = \frac{0.012 \times 21.0 \times 350 \times 55^2}{27.5^3} \times \frac{1.2}{1.2} = 12.8 \text{ Pa}$$

$$P = \frac{0.012 \times 21.0 \times 350 \times 165^2}{27.5^3} \times \frac{1.2}{1.2} = 115 \text{ Pa}$$

- 4.10.5 [Q] Calculate the velocity (m/s) and pressure loss (Pa) for air quantities of 60 m³/s, 125 m³/s, 190 m³/s & 250 m³/s in a vertical raisebored shaft 350 m long and 4.0m diameter. State your assumptions.

- 4.10.5 [A] Assumptions: K Factor = 0.0045 Ns²/m⁴ Density of air 1.2 kg/m³
Determine area (A) and circumference (C) for raisebored shaft

Diameter (m)	C (m)	A (m ²)
4.0	12.5664	12.5664

Calculate velocity $v = \frac{Q}{A}$

$$v = \frac{60}{12.5664} = 4.8 \text{ m/s}$$

$$v = \frac{125}{12.5664} = 9.9 \text{ m/s}$$

$$v = \frac{190}{12.5664} = 15.1 \text{ m/s}$$

$$v = \frac{250}{12.5664} = 19.9 \text{ m/s}$$

Calculate pressure: $P = \frac{kCLQ^2}{A^3} \times \frac{\rho}{\rho_s}$

$$P = \frac{0.0045 \times 12.5664 \times 350 \times 60^2}{12.5664^3} \times \frac{1.2}{1.2} = 36 \text{ Pa}$$

$$P = \frac{0.0045 \times 12.5664 \times 350 \times 125^2}{12.5664^3} \times \frac{1.2}{1.2} = 156 \text{ Pa}$$

$$P = \frac{0.0045 \times 12.5664 \times 350 \times 190^2}{12.5664^3} \times \frac{1.2}{1.2} = 360 \text{ Pa}$$

$$P = \frac{0.0045 \times 12.5664 \times 350 \times 250^2}{12.5664^3} \times \frac{1.2}{1.2} = 623 \text{ Pa}$$

- 4.10.6 [Q] Determine the pressure losses contributed by the shock in a 90° bend of a blasted rock airway 5.0m high and 5.0m wide for air quantities of 20 m³/s, 75m³/s 125 m³/s. Assume standard air density.

- 4.10.5 [A] Shock factor X for 90° rectangular bend = 1.15
Determine area of drive = h x w = 5 x 5 = 25.0 m²

Calculate velocity in drive: $v = \frac{Q}{A}$

$$v = \frac{20}{25.0} = 0.8 \text{ m/s}$$

$$v = \frac{75}{25.0} = 3.0 \text{ m/s}$$

$$v = \frac{125}{25.0} = 5.0 \text{ m/s}$$

Calculate velocity pressure $VP = \frac{\rho v^2}{2} \text{ (Pa)}$

$$VP = \frac{1.2 \times 0.8^2}{2} = 0.4 \text{ Pa}$$

$$VP = \frac{1.2 \times 3.0^2}{2} = 5.4 \text{ Pa}$$

$$VP = \frac{1.2 \times 5.0^2}{2} = 15.0 \text{ Pa}$$

Calculate shock loss $P_{\text{SHOCK}} = XVP$

$$P_{\text{SHOCK}} = 1.15 \times 0.4 = 0.46 \text{ Pa}$$

$$P_{\text{SHOCK}} = 1.15 \times 5.4 = 6.2 \text{ Pa}$$

$$P_{\text{SHOCK}} = 1.15 \times 15 = 17.25 \text{ Pa}$$

- 4.10.7 [Q] Determine the pressure losses contributed by the shock in a 90° smooth bend with a centre line radius of 15 m in a 1.2m diameter flexible duct with an air quantity of m³/s. Assume standard air density.

- 4.10.7 [A] Shock factor X for 90° circular bend $= X = \frac{0.25}{m^2 a^{0.5}} \times \frac{\theta^2}{90^2}$

$$m = \frac{20}{1.2} = 16.6667$$

$$a = \frac{1.2}{1.2} = 1$$

$$X = \frac{0.25}{16.6667^2 \times 1^{0.5}} \times \frac{90^2}{90^2} = 0.0009$$

Determine area of duct $= \pi r^2 = \pi \times 0.6^2 = 1.1310 \text{ m}^2$

Calculate velocity in drive: $v = \frac{Q}{A}$

$$v = \frac{30}{1.1310} = 26.5 \text{ m/s}$$

Calculate velocity pressure $VP = \frac{\rho v^2}{2}$ (Pa)

$$VP = \frac{1.2 \times 26.5^2}{2} = 422 \text{ Pa}$$

Calculate shock loss $P_{\text{SHOCK}} = XVP$

$$P_{\text{SHOCK}} = 0.0009 \times 422 = 0.38 \text{ Pa}$$

- 4.10.8 [Q] Determine the equivalent airway length of shock losses in a 90° bend in a blasted rock airway 5.0m high and 5.0m wide for velocities of 1.0 m/s, 2.5 m/s, 5.0 m/s, 7.5m/s & 10.0 m/s. Assume standard air density

4.10.8 [A] $Le = \frac{XD_h}{6.67k}$ $Le = \frac{1.15 \times 5}{6.67 \times 0.012}$ 71.8m and will be the same irrespective of flow.

- 4.10.9 [Q] Determine the pressure loss when an airway reduces abruptly in size from 6.0m high and 6.0m wide to an airway 3.0m high and 3.0m wide for air quantities of 10 m³/s, 50 m³/s & 100 m³/s. Assume standard air density.

- 4.10.9 [A] Abrupt Contraction $X = 0.5 (1 - (A_2/A_1))$

$$A_1 = 6 \times 6 = 36 \text{ m}^2 \quad A_2 = 3 \times 3 = 9 \text{ m}^2$$

$$X = 0.5 (1 - (9/36)) = 0.5 (1 - 0.25) = 0.375$$

Pressure loss occurs in the entry airway

$$\text{Velocity in } A_1 = \frac{10}{36} = 0.2778 \text{ m/s and } VP = \frac{1.2 \times 0.2778^2}{2} = 0.0463 \text{ Pa}$$

$$\text{Therefore } P_{\text{SHOCK}} = 0.375 \times 0.0463 \text{ Pa}$$

$$\text{Velocity in } A_1 = \frac{50}{36} = 1.3889 \text{ m/s and } VP = \frac{1.2 \times 1.3889^2}{2} = 1.1574 \text{ Pa}$$

$$\text{Therefore } P_{\text{SHOCK}} = 0.375 \times 1.1574 = 0.4340 \text{ Pa}$$

$$\text{Velocity in } A_1 = \frac{100}{36} = 2.7778 \text{ m/s and } VP = \frac{1.2 \times 2.7778^2}{2} = 4.6296 \text{ Pa}$$

$$\text{Therefore } P_{\text{SHOCK}} = 0.375 \times 4.6296 = 1.7361 \text{ Pa}$$

- 4.10.10 [Q] Determine the pressure loss when an airway increases abruptly in size from 6.0m high and 6.0m wide to an airway 3.0m high and 3.0m wide for air quantities of 10 m³/s, 50 m³/s & 100 m³/s. Assume standard air density.

- 4.10.10 [A] Abrupt Expansion $X = (1 - (A_1/A_2))^2$

$$A_1 = 3 \times 3 = 9 \text{ m}^2 \quad A_2 = 6 \times 6 = 36 \text{ m}^2$$

$$X = (1 - (9/36))^2 = (1 - 0.25)^2 = 0.5625$$

Pressure loss occurs in the entry airway

$$\text{Velocity in } A_1 = \frac{10}{9} = 1.1111 \text{ m/s and } VP = \frac{1.2 \times 1.1111^2}{2} = 0.7407 \text{ Pa}$$

$$\text{Therefore } P_{\text{SHOCK}} = 0.5625 \times 0.7407 = 0.4167 \text{ Pa}$$

$$\text{Velocity in } A_1 = \frac{50}{9} = 5.5556 \text{ m/s and } VP = \frac{1.2 \times 5.5556^2}{2} = 18.5185 \text{ Pa}$$

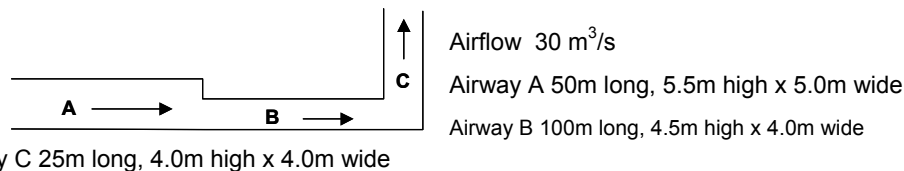
$$\text{Therefore } P_{\text{SHOCK}} = 0.5625 \times 18.5185 = 10.4167 \text{ Pa}$$

$$\text{Velocity in } A_1 = \frac{100}{9} = 11.1111 \text{ m/s and } VP = \frac{1.2 \times 11.1111^2}{2} = 74.0739 \text{ Pa}$$

$$\text{Therefore } P_{\text{SHOCK}} = 0.5625 \times 74.0739 = 41.6666 \text{ Pa}$$

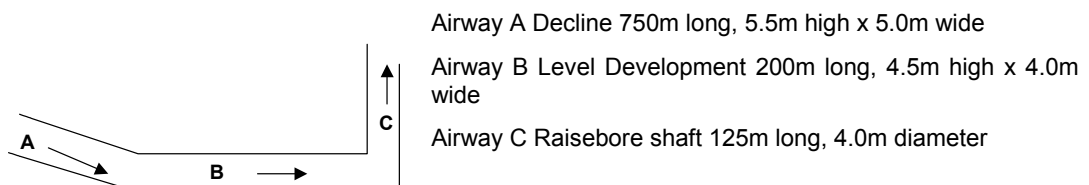
- 4.10.11 [Q] Calculate the frictional pressure loss for the following systems. State your assumptions for Atkinsons factors. Assume standard air density.

i)



ii)

Airflow 100 m³/s



- 4.10.11 1) Assume friction factor 0.012 Ns²/m⁴ for rock walled air way

[A]

Determine the resistance for each airway. $R = \frac{kCL}{A^3} \text{ Ns}^2/\text{m}^8$

$$\text{Airway A Area} = 27.5 \text{ m}^2 \text{ Circumference} = 21 \text{ m and } R_A = \frac{0.012 \times 21 \times 50}{27.5^3} = 0.0006 \text{ Ns}^2/\text{m}^8$$

$$\text{Airway B Area} = 18 \text{ m}^2 \text{ Circumference} = 17 \text{ m and } R_B = \frac{0.012 \times 17 \times 100}{18^3} = 0.0035 \text{ Ns}^2/\text{m}^8$$

$$\text{Airway C Area} = 16 \text{ m}^2 \text{ Circumference} = 16 \text{ m and } R_C = \frac{0.012 \times 16 \times 25}{16^3} = 0.0012$$

$$\text{Ns}^2/\text{m}^8$$

$$R_T = R_A + R_B + R_C = 0.0006 + 0.0035 + 0.0012 = 0.0053 \text{ Ns}^2/\text{m}^8$$

Calculate frictional pressure loss

$$P = RQ^2 = 0.0053 \times 30^2 = 5 \text{ Pa}$$

- 2) Assume friction factor $0.012 \text{ Ns}^2/\text{m}^4$ for rock walled air way and $0.0045 \text{ Ns}^2/\text{m}^4$ for raise bored airway

Determine the resistance for each airway. $R = \frac{kCL}{A^3} \text{ Ns}^2/\text{m}^8$

Airway A Area = 27.5 m^2 Circumference = 21 m and

$$R_A = \frac{0.012 \times 21 \times 750}{27.5^3} = 0.0091 \text{ Ns}^2/\text{m}^8$$

Airway B Area = 18 m^2 Circumference = 17 m and

$$R_B = \frac{0.012 \times 17 \times 200}{18^3} = 0.0070 \text{ Ns}^2/\text{m}^8$$

Airway C Area = 12.56 m^2 Circumference = 12.56 m and

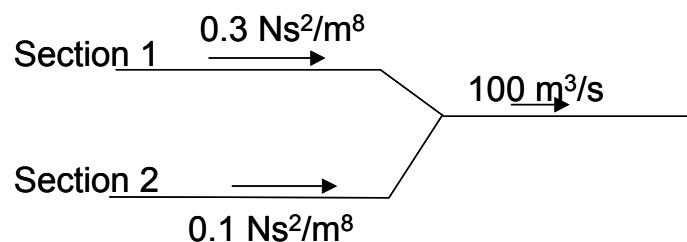
$$R_A = \frac{0.0045 \times 12.56 \times 125}{12.56^3} = 0.0036 \text{ Ns}^2/\text{m}^8$$

$$R_T = R_A + R_B + R_C = 0.0091 + 0.0070 + 0.0036 = 0.0197 \text{ Ns}^2/\text{m}^8$$

Calculate frictional pressure loss

$$P = RQ^2 = 0.0197 \times 100^2 = 197 \text{ Pa}$$

- 4.10.12 Two sections of a mine are depicted in the following figure
[Q]



1. Determine the airflow through each section of the mine
2. Calculate the pressure drop necessary to achieve $100 \text{ m}^3/\text{s}$.

- 4.10.12 Section 1 is in parallel with section two and the pressure is the same across both sections of the mine.
[A]

Calculate the resistance $\frac{1}{\sqrt{R_T}} = \frac{1}{\sqrt{R_{\text{SECTION1}}}} + \frac{1}{\sqrt{R_{\text{SECTION2}}}}$

$$\frac{1}{\sqrt{R_T}} = \frac{1}{\sqrt{0.3}} + \frac{1}{\sqrt{0.1}} = 1.8257 + 3.1623 = 4.9880$$

$$R_T = \frac{1}{4.9880^2} = 0.0402 \text{ N s}^2/\text{m}^8$$

As $P = RQ^2$ and since P is constant $R_T Q_T^2 = R_1 Q_1^2$

The airflow in Section 1 $0.0402 \times 100^2 = 0.3 \times Q_1^2$

$$Q_1^2 = \frac{0.0402}{0.3} \times 100^2 = 1,3387.75$$

$$Q_1 = \sqrt{13387.75} = 36.6 \text{ m}^3/\text{s}$$

Similarly the airflow in Section 2 is $0.0402 \times 100^2 = 0.1 \times Q_1^2$

$$Q_2 = 63.4 \text{ m}^3/\text{s}.$$

$$\begin{aligned} \text{Pressure is } P &= RQ^2 \\ &= 0.0402 \times 100^2 \\ &= 402 \text{ Pa} \end{aligned}$$

- 4.10.13 [Q] The raise bored airway for a mine has become unstable and must be sprayed with shotcrete to prevent it collapsing. The shaft is 3.2m diameter and 200 m in length. The concrete will be applied to a thickness of 100mm. The current airflow is 90 m³/s. What will the airflow be once the shotcreting has been completed?. Assume the same pressure drop.

- 4.10.13 [A] Determine the existing resistance. $R = \frac{kCL}{A^3} \text{ N s}^2/\text{m}^8$ Diameter = 3.2m

$$R = \frac{0.0045 \times 10.05 \times 200}{8.04^3} = 0.0174 \text{ N s}^2/\text{m}^8$$

Determine the new resistance. $R = \frac{kCL}{A^3} \text{ N s}^2/\text{m}^8$ Diameter = 3.2 – (2 x 0.100) = 3.0m

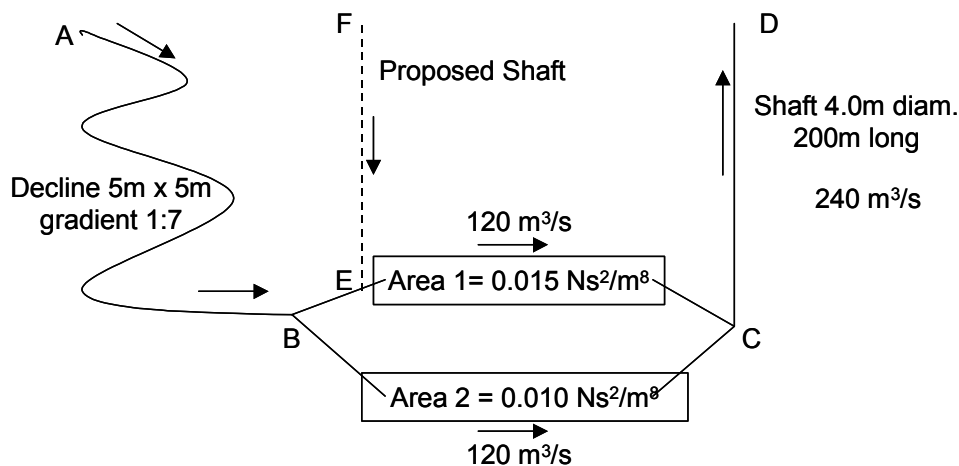
$$R = \frac{0.0045 \times 9.42 \times 200}{7.07^3} = 0.0240 \text{ N s}^2/\text{m}^8$$

Since pressure is constant $R_1 Q_1^2 = R_2 Q_2^2$

$$Q_2^2 = \frac{0.0174}{0.0240} \times 90^2$$

$$\begin{aligned} Q_2 &= \sqrt{5872.5} \\ &= 76.6 \text{ m}^3/\text{s} \end{aligned}$$

- 4.10.14 [Q] The figure below represents the workings of an underground mine.



It is proposed to develop a new shaft (FE) and increase the mine airflow from 125 m³/s up to 240 m³/s.

It is also necessary to provide equal air to each section of the mine.

What diameter should the new shaft be? Assume the new shaft can be raisebored.

4.10.14
[A]

Determine the resistance of the decline. $R = \frac{kCL}{A^3} \text{ Ns}^2/\text{m}^8$

At 1:7 the decline will be $200 \times 7 = 1,400 \text{ m}$ long Area = 25 m^2 an Circumference = 20 m .

Assume a friction factor of the decline = $0.012 \text{ Ns}^2/\text{m}^4$

$$R_{AB} = \frac{0.012 \times 20 \times 1400}{25^3} = 0.0215 \text{ Ns}^2/\text{m}^8$$

Determine the resistance of A,B, Area 2,C

$$R = 0.0215 + 0.0100 = 0.0315 \text{ Ns}^2/\text{m}^8$$

For an equal split the resistance in both sections must be equal

$$R_{AB \text{ Area } 2 \text{ C}} = R_{FE \text{ Area } 1 \text{ C}}$$

$$\text{Therefore } R_{FE} = R_{AB \text{ Area } 2 \text{ C}} - R_{\text{Area } 1} = 0.0315 - 0.0150 = 0.0165 \text{ Ns}^2/\text{m}^8$$

Calculate the diameter

Area = π

$$\text{Area} = \frac{\pi D^2}{4} \text{ and Circumference} = \pi D$$

$$\text{Substituting in } R = \frac{kCL}{A^3} \quad R = \frac{k\pi DL}{\frac{\pi D^2}{4}^3}$$

Solving for D

$$D = (4/R^{1/3} \pi)^{3/5} (k \pi L)^{1/5}$$

$$D = (4 / 0.0165^{1/3} \pi)^{3/5} (0.0045 \pi 200)^{1/5}$$

$$D = 3.2339\text{m diameter}$$

“Reason, Observation and Experience – the Holy Trinity of Science.”

Robert G. Ingersoll (1833-99) US Lawyer and Agnostic. The Gods.

5 MEASURING AIRFLOW

Airflow is measured in units of cubic metres per second (m^3/s). To measure airflow, the airway cross-sectional area and air velocity need to be known.

There are a large variety of air velocity measuring instruments, however the following are most applicable to mining requirements:

- Vane Anemometer
- Hot Wire Anemometer
- Velometer
- Smoke Tube

5.1 Vane Anemometer

The instrument consists simply of a light impeller that rotates at a speed dependent on flow rate. The impeller rotational speed is converted into an air velocity reading using either electronic or mechanical means. Most anemometers used in mine ventilation survey work are capable of measuring velocities in the range of 0.2 to 25 m/s, with an absolute accuracy of ± 0.5 m/s, or better. They are ideally suited to conducting primary ventilation surveys and for routine determination of air velocity. Air velocities below 0.2 m/s should be measured with another technique such as a smoke tube or hot wire anemometer.



Mechanical anemometers (particularly the Lambrecht 1406a) are used in the majority of Australian mines. The mechanical instruments are preferred to the electronic ones for the following reasons:

- They automatically engage their clutch to commence measurement about five seconds after the mechanical start lever is depressed. This gives the operator sufficient time to move the instrument away from the body to the position where the traverse will be commenced.
- When they are fitted with a timing mechanism, they can integrate velocity readings over a fixed time period (usually one minute), making them ideally suited to traversing measurement techniques. Traversing is the fastest way of averaging air velocity over an airway cross-sectional area, with accuracy sufficient for a primary ventilation survey. Electronic instruments will generally only provide spot readings. Some can integrate, but this requires a switch to be held down. This can often be difficult to accomplish when the instrument is mounted on a pole.
- Mechanical instruments are usually smaller and easier to carry. They seem to also be more reliable and robust than their electronic counterparts.

Mechanical anemometers are delicate instruments and must be treated with care. They should always be transported in the instrument case. Recalibration can be organised through the CSIRO or the supplier and should be undertaken at least annually.

5.2 Hot Wire Anemometer

The hot wire anemometer consists of an electrically heated wire at the end of an extendible wand. In the most popular type of instrument, the wire is maintained at a constant temperature. The amount of current required to do this is related to the speed of the air. The instrument contains a second internal hot wire to provide compensation for changes in air temperature.

Hot wire anemometers are very useful for accurately measuring low air velocities (down to about 0.05 m/s). Before making a velocity measurement, it is important that the flow distribution and direction be checked first, using smoke tubes.

Hot wire anemometers are generally not designed for use in the traversing measurement technique. They are suitable for taking spot measurements and most instruments have the facility to electronically average a number of spot readings. They are generally not directionally sensitive.

5.3 Velometer

The velometer is a mechanical instrument that consists simply of a hinged vane that swings in a rectangular passage. A port in the side of the instrument casing is connected to the passage. When the instrument is held in an airstream, airflow through the port deflects the vane against spring pressure. The vane is connected directly to a velocity indication scale. Pitot type tubes can be attached to the port with hosing to enable velocity to be checked in ducts.

This instrument provides fast, accurate measurements of velocities down to about 0.2 m/s. It is well suited to routine work such as determining face velocities in secondary ventilation surveys.

The velometer is directionally sensitive and can only be used for spot surveys. It must be read while being held by the observer and care must be taken to hold the instrument sufficiently far from the observer to avoid interference with the reading. Velometers are not widely used in Australian mines.

5.4 Smoke Tube

A smoke tube consists of a glass tube (similar to a gas detector tube) that emits a highly visible smoke when air is aspirated through it via a rubber bulb. Smoke tubes give superior results to other less satisfactory improvised alternatives such as spray paint, a handful of fine dry dust or, cigarette smoke!

Measurements are taken by marking out a fixed distance in a drive with two paint lines on the roadway. The distance should be that which the smoke travels in about 30 seconds (the smoke cloud usually dissipates after this period of time). Smoke tubes are used for velocities of less than 0.1 m/s to 0.5 m/s and a measurement distance of about 5m is usually therefore appropriate.

The smoke is released in the centre of the airway about an arm's length upstream of the first mark. Two observers are required to determine the time interval between when the centre of the smoke cloud passes the first and second mark. The distance that the smoke travelled divided by the time taken, then multiplied by a correction factor (typically 0.8) calculates the velocity of the air. The correction factor allows for the effects of smoke dispersal and the fact that the velocity in the centre of the drive is usually higher than the average velocity.

It is always a good idea to check the flow distribution in the airway before commencing a smoke tube measurement. In hot, inclined airways with low velocities, the magnitude and even direction at the backs may be different to that on the floor.

Smoke tubes can also be used to trace air leakage (e.g. into draw-points) and to help visualise the flow distribution.

5.5 Pitot-Static Tube

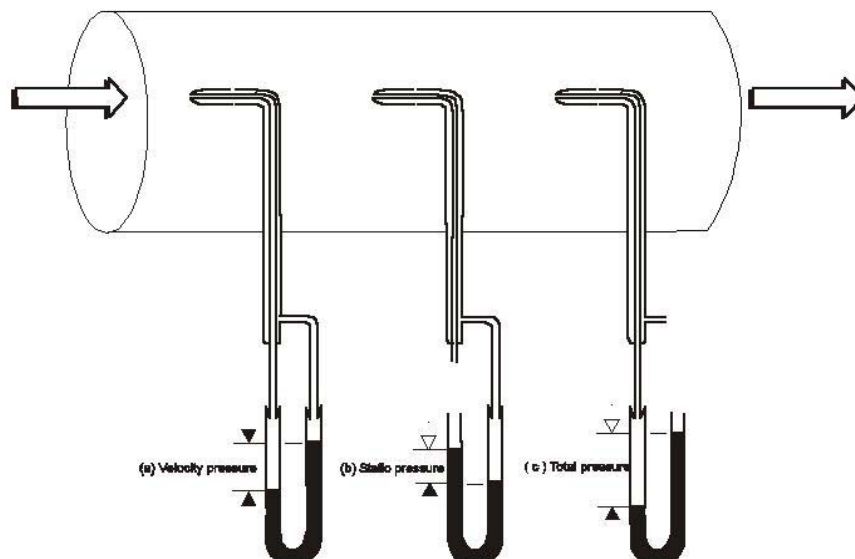
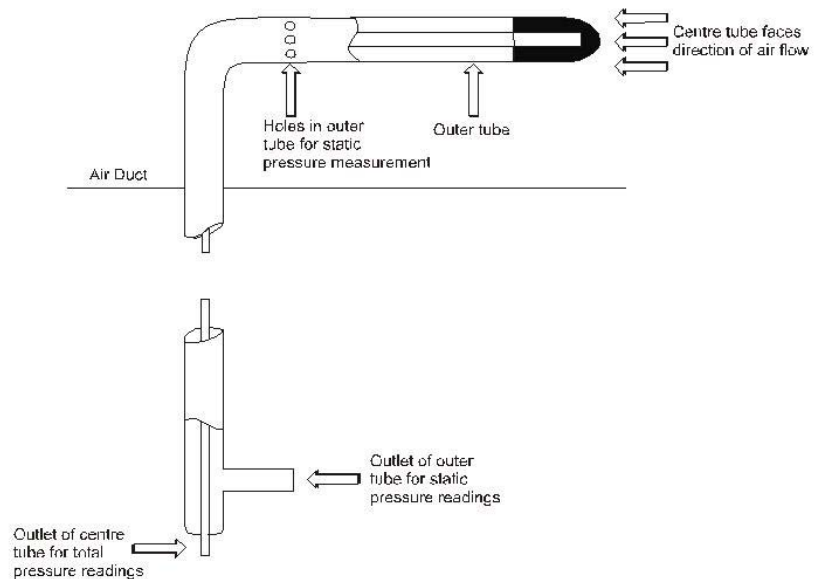
The Pitot-static tube (often referred to as the Pitot Tube) consists of two concentric "L" shaped tubes.

A pitot static tube is used to determine pressure at a point. A variety of designs have been developed with advantages between the different designs generally applicable to research work.

The pitot static tube can be used successfully without the need for corrections given if:

- The head is aligned to the direction of airflow within 15° (2.5% error in the velocity).
- The tube diameter must not exceed $1/25$ the duct diameter in circular ducts and $1/20$ the length of the smallest side in a rectangular duct.
- The tube diameter should not exceed 15 mm.
- The velocity should be between 3m/s and 70m/s.

The pitot-static tube can be used to measure static, total and velocity pressure when connected to a manometer.



With an accurate manometer, good measurement technique and steady flow conditions, velocities can be measured to within $\pm 1\%$ for velocities above 5 m/s. The accuracy below 5 m/s is limited by the inability of most manometers to resolve very low pressures (less than 10 Pa).

Because Pitot-static tubes are essentially a medium - high velocity measurement instrument, they are not used for surveys in underground drives. They are most useful for the measurement of the higher air velocities often found in ducts. Typically, a pitot-static tube would be used for fan testing and flow measurement in secondary ventilation ducts. The most useful length for many mines is 1.5m.

Of the above instruments, a vane anemometer, smoke tube set and pitot-static tube are considered essential for any underground mine.

Summary - Air Velocity Measurement Instruments

Instrument	Velocity Range (m/s)	Comments
Vane Anemometer	0.2 - 25 *	Versatile and accurate - the most useful ventilation instrument – particularly for ventilation surveys. Mechanical type is preferred.
Hot Wire Anemometer	0.05 - 15	Good for accurate spot measurement of low velocities
Velometer	0.2 - 15	Useful for quick, routine spot air velocity measurement. Method not accurate enough for ventilation surveys.
Smoke Tube	0.05 - 0.5	Enables air flow distribution to be "seen". Cumbersome method to use
Pitot-Static Tube	5 - 50	Best instrument for measuring airflow in ducts.

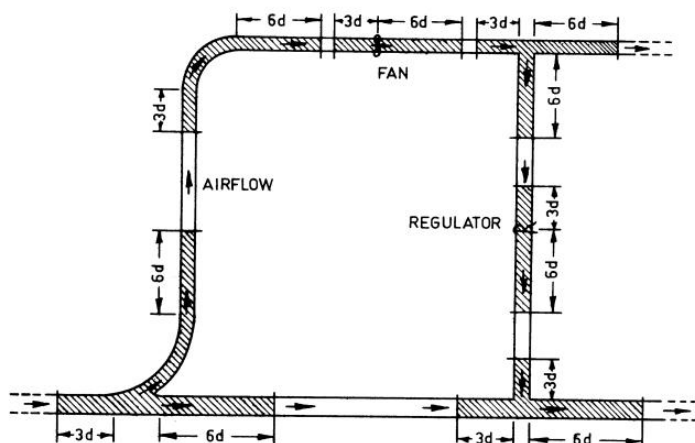
* Instruments capable of reading up to 50 m/s are available, but not generally required for mine ventilation work.

5.6 Selection of Measurement Site

The ideal airflow-measuring site will have a 'smooth' or 'undisturbed' airflow distribution pattern. This is will be an area in a drive that is:

- as straight as possible,
- of uniform shape and size and,
- unobstructed.

As a rule of thumb, the measurement site should be located at least six drive diameters downstream and a minimum of three drive diameters upstream of any disturbing influence (bend, obstruction, etc).



This is often not possible in an underground operation, and a compromise is necessary. In many cases this will mean taking extra measurements at site that better fit the selection criteria.

5.7 Measuring the Airway Cross Sectional Area

Several methods are available to determine ventilation station area. When conducting ventilation surveys that require a high level of accuracy the mine surveyors should be utilised to accurately determine the shape and area of all permanent ventilation stations.

For primary ventilation surveys, permanent ventilation stations must be clearly marked with a paint line on the walls and backs. The station number and surveyed cross-sectional area should also be marked. The most reliable method is to write this information on a steel or plastic tag that is attached to the wall using surveyor's putty.

For temporary ventilation stations a tape measure is required to determine a minimum of five *equally* spaced heights (offsets). Using these measurements the area can be calculated using **Simpson's Rule**.

"The area enclosed by a curvilinear figure divided into an even number of strips of equal width is equal to one-third of the width of the strip, multiplied by the sum of the two extreme offsets, twice the sum of the remaining odd offsets, and four-times the sum of the even offsets."

$$\text{Area} = \frac{X}{3} (A + 2O + 4E)$$

Where

X = distance between offsets. (m)

A = sum of the first and last off sets. (m)

O = sum of the remaining odd offsets. (m)

E = sum of the even offsets. (m)

For example

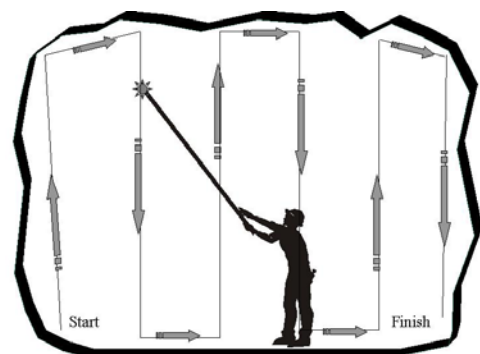
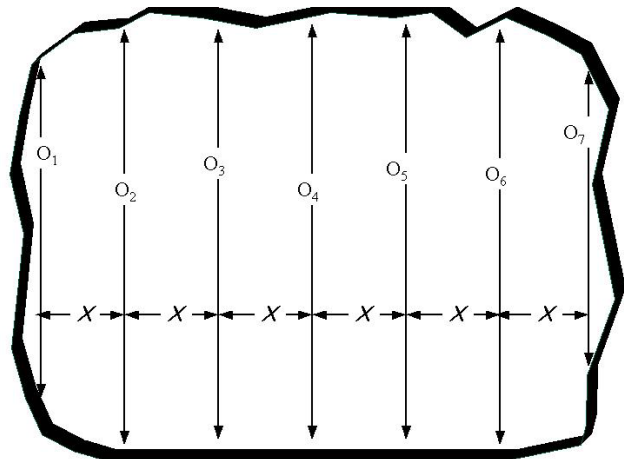
$$\text{In this case Area} = \frac{X}{3} \times [(O_1 + O_7) + 2(O_3 + O_5) + 4(O_2 + O_4 + O_6)]$$

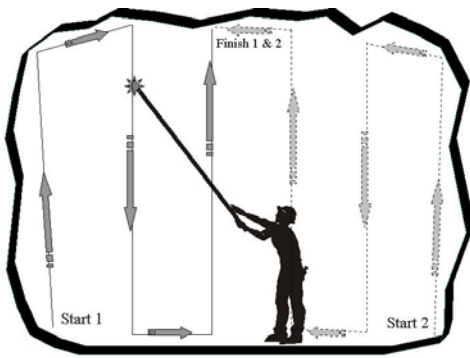
Note that the area is divided into an even number of sections i.e. an odd number of offsets are measured.

It is very important to note that in trackless mining, the ventilation station cross-sectional areas will change over time. This is mainly due to extra road base material being applied. In one mine, it was found that over about 10 years, the cross-sectional areas of a number of ventilation stations had decreased by over 10%. In recognition of this change ventilation stations maintained in main access declines should be remeasured at least every two years.

5.8 Traverse Velocity Measurement

The traverse method is the quickest way to determine air velocity with sufficient accuracy for a ventilation survey. An integrating anemometer is required for this technique and the vane anemometer is generally the only instrument with this facility.





The principle of the traverse technique is to slowly traverse the anemometer across the airway at a uniform speed and in a pattern that ensures that the entire airway cross-sectional area is evenly represented. In order to ensure acceptable accuracy, traverse speed should be limited to less than 0.4 m/s (less than 0.2 m/s when measuring air velocities below 0.75 m/s).

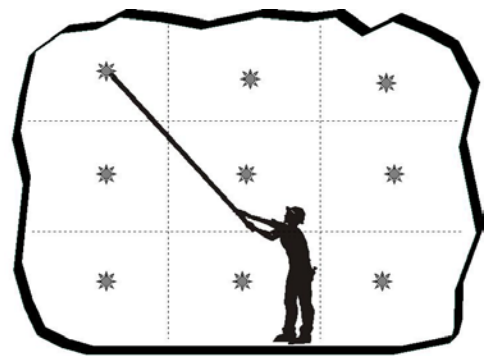
When using an anemometer with a 60 second timer two readings will be required to complete one traverse – one for each side of the opening, and the velocity for the area will be the average of both of these readings). Multiple

traverses readings will be undertaken until a minimum of two traverse sets agree to within $\pm 5\%$ of one another.

5.9 Spot Reading Method

For spot readings, the airway cross-section is subdivided into a number of equal areas. A velocity reading is taken in the centre of each of these smaller areas. The velocity for the area will be the average of all spot readings.

It should be noted that for acceptable accuracy, the size of the squares should not be significantly larger than 1m x 1m. This is a precise, but also time-consuming method. Because of this, it is not commonly used for routine work.



5.10 Single Spot Reading Method

A simple, quick, and rough form of spot reading is to take a single velocity measurement in the centre of a drive, or duct. This reading will almost always over-estimate the average velocity, because the fastest velocity is usually found in the centre of the airway. A correction factor (often between 0.8 and 0.9) therefore needs to be applied. This method is useful for routine work, but is not accurate enough for a primary ventilation survey. It also requires smooth, unobstructed flow conditions in the airway.

“I will now speak of ventilating machines. If the shaft is very deep and no tunnel reaches to it, or no drift from another shaft connects with it, or when a tunnel is of great length and no shaft reaches it, then the air does not replenish itself. In such a case it weighs heavily on the miners, causing them to breath with difficulty and some times they are even suffocated and burning lamps are also extinguished. There is therefore a necessity for machines which enable the miners to breathe easily and carry on their work”

Agricola “De Re Metallica” (1556)

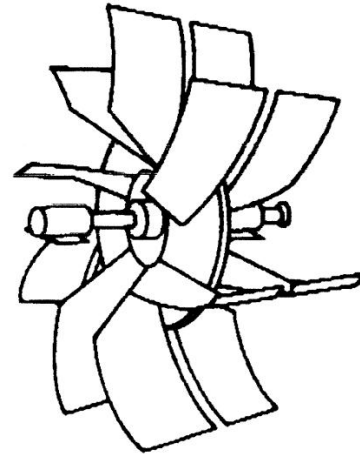
6 UNDERSTANDING FANS

In Chapter 3 we discussed the effect of natural ventilating pressure and how it can cause air to flow through mine workings. Airflow through the large mechanised mining operations of today, could not exist in the quantities required, unless there was sufficient pressure difference to overcome the resistances in the circuit. This requires mechanical energy input provided by fans.

The first evidence of mechanical methods for ventilating mine were described by Agricola in 1556 and involved methods such as human powered centrifugal fans, bellows connected to wooden conduits and shaft collar deflectors, which divert winds into the mine workings.

Although large centrifugal fans were in use in the early to mid 1850's, airflow through the underground workings was generally provided only by natural draft. It was not until the turn of the 20th century that fans were used extensively in mines.

Typically, these impellers (pictured opposite) were fitted with backward curved paddle blades, between 240 inch (6.0m) and 520 inch (13.0m) diameter and rotating at very slow speeds between 0.67 revolutions per second (rps) (40 rpm) and 1.0 rps (60 rpm) and, driven by large steam powered engines. One of the early makes of this type of fan was the double inlet Walker Indestructible¹⁶.



In around 1900, Samuel Davidson developed a forward curved multi bladed fan to handle hot air for the purpose of drying tealeaves from their Colombo plantations in Ceylon, now Sri Lanka. The fan impeller was much smaller in diameter with forward curved blades designed to run at significantly higher speeds. The fan was called a Sirocco after the hot winds that blow across the Mediterranean from the coast of North Africa.

¹⁶ de la HARPE, J.H., JENNER, L.W. “The History of Mine Fans in South Africa” The Journal of the Mine Ventilation Society of South Africa. (December 1986)



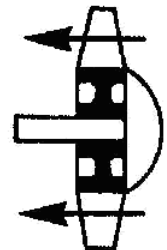
It was one of these new generation fans that are believed to be the first mechanical means of ventilation installed in Australia. This fan was installed around 1917 at on No. 4 Shaft Zinc Mine Broken Hill and had an airflow capacity of 45,000 cfm ($21.0\text{m}^3/\text{s}$) and remained in operation at this location until it was replaced in 1921. The replacement was a larger Sirocco fan (pictured left), which was manufactured in Belfast Ireland by Davidson & Co. Ltd. This fan was fitted with a 119-inch (3.0m) impeller designed to produce 195,200 cfm ($92\text{m}^3/\text{s}$) at 6 of inches water gauge (1495 Pa) when rotating at 244 rpm.

6.1 General Description of Fans

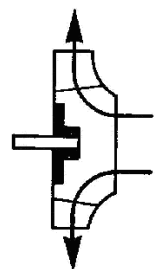
A fan is a rotating machine in which air is continuously drawn in at one pressure and delivered at a higher pressure i.e. the mechanical energy delivered to the fan is transformed into potential energy (pressure) and kinetic energy (velocity) by the fan. This pressure is necessary to overcome the resistance of the particular duct or mine, in which the fan is operating.

Fans are named according to the type of impeller fitted into the casing. In mining applications the main fan types are:

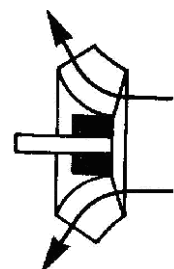
1. **Axial-flow.** - In an axial fan the air flows through the impeller parallel to and at a constant distance from the axis. The pressure rise is provided by the direct action of the blades.



2. **Centrifugal or Radial-flow.** - In centrifugal fans the air enters parallel to the axis of the fan turns through 90° and is discharged radially through the blades. The blade force is tangential causing the air to spin with the blades and the main pressure rise is attributed to the centrifugal force.



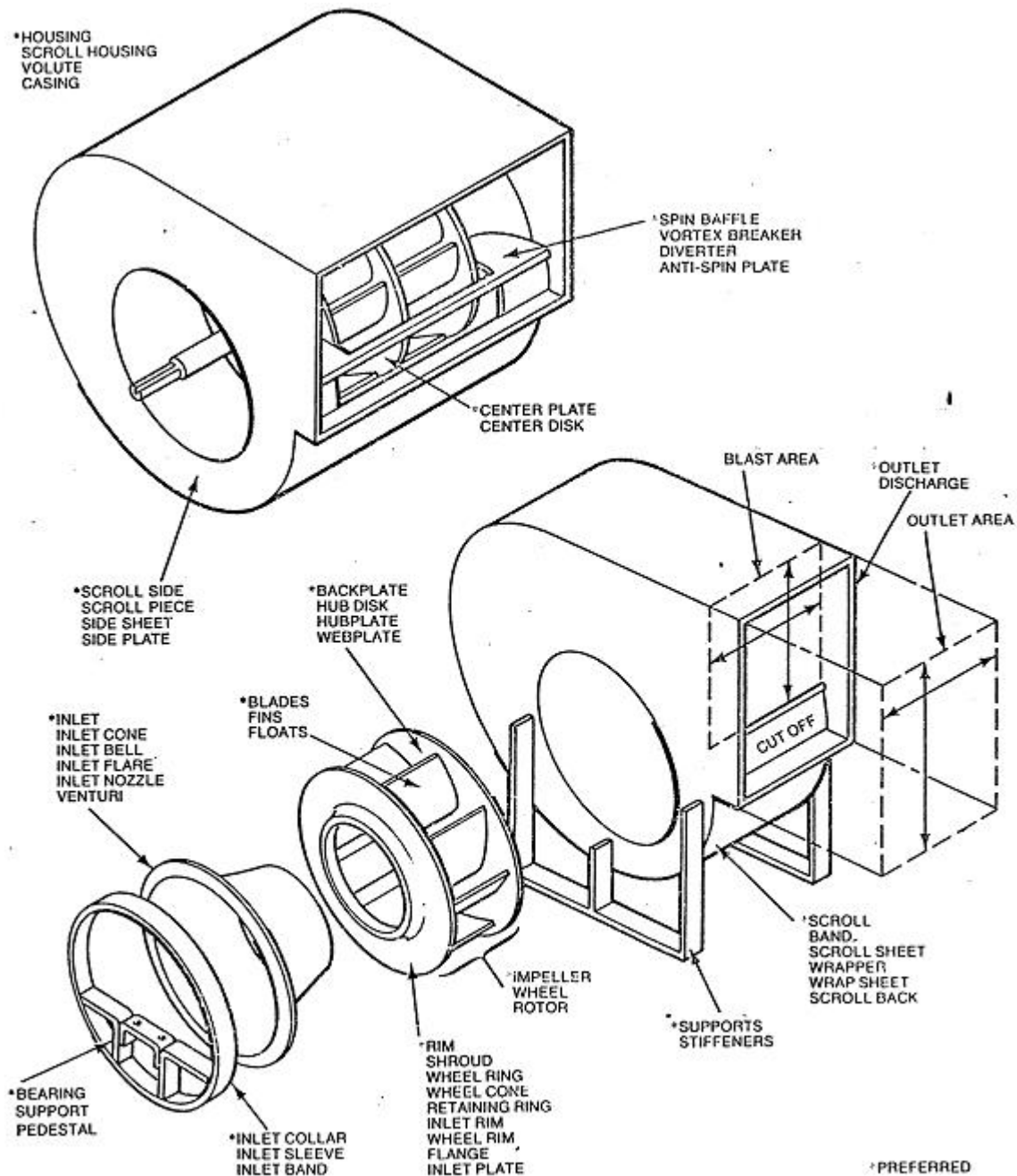
3. **Mixed flow.** - In a mixed flow fan the air enters parallel to the axis of the fan turns through an angle which may range from 30° to 90° . The pressure rise is partially by direct blade action and partially by centrifugal action.



6.2 Fan Terminology

The principal component of a fan is the impeller, consisting of the hub and blades. The impeller is mounted on a shaft inside the housing, or casing, of the fan that is driven directly or indirectly by a motor. In addition, the fan is generally equipped with an inlet cone and an outlet evasè to improve the flow characteristics of the fan. There may also be guide vanes to control the direction of airflow through the fan.

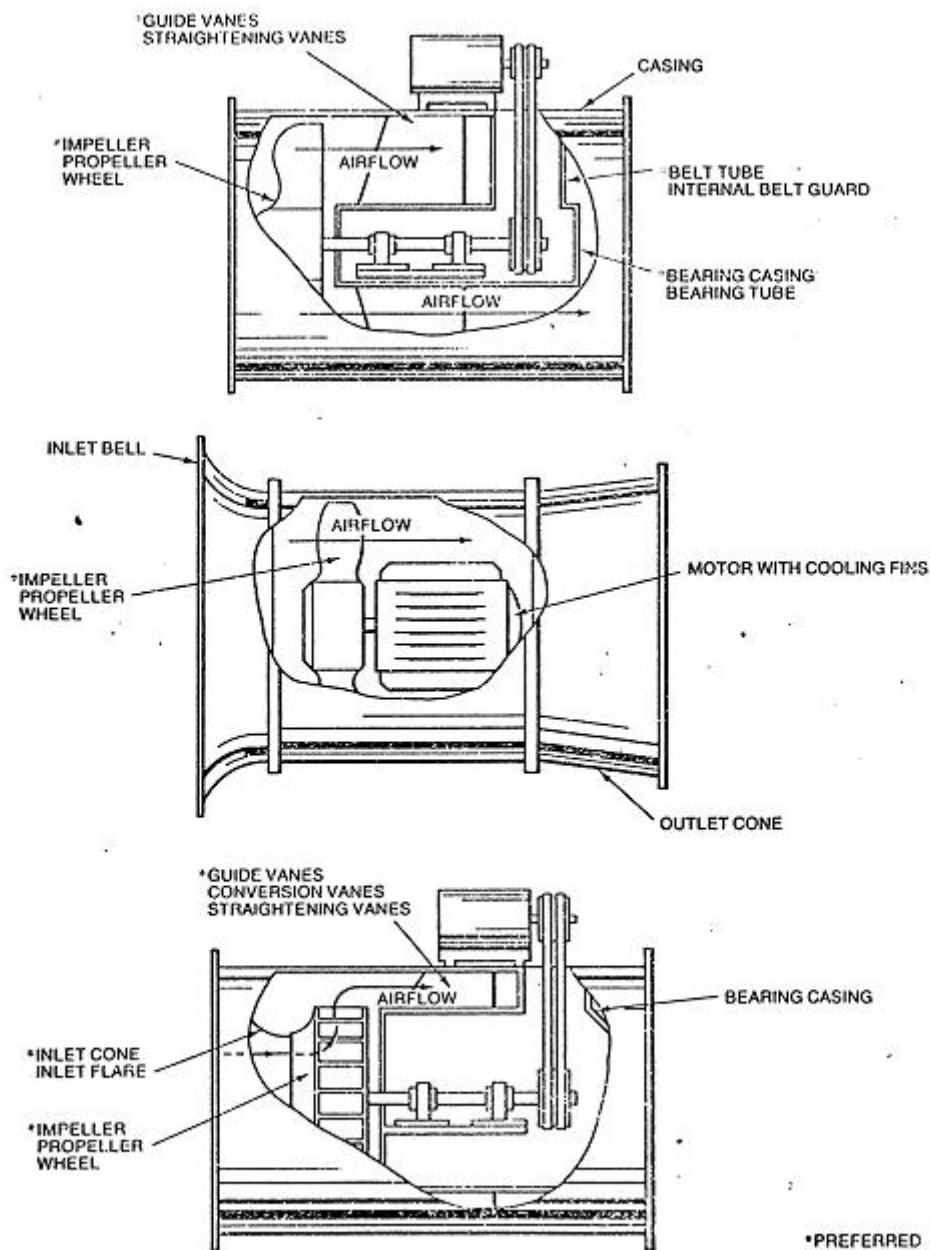
Centrifugal



COMMON TERMINOLOGY FOR CENTRIFUGAL FAN COMPONENTS

Reproduced from Air Moving and Conditioning Association Inc. "AMCA Fan Application Manual"

Axial



COMMON TERMINOLOGY FOR AXIAL AND TUBULAR CENTRIFUGAL FANS

Reproduced from Air Moving and Conditioning Association Inc. "AMCA Fan Application Manual"

6.3 Axial-Flow Fans

Axial flow fans are divided into three categories

1. Free air - The impeller is not confined e.g. Desk fan
2. Tube axial - The impeller is confined in a casing

3. Vane axial - Same as a tube axial except guide vanes are fitted inside the casing to reduce swirl

The impeller may have two or more blades. Generally speaking, high-pressure fans have a greater number of blades than low-pressure fans. The shape and pitch of the blades vary considerably in different designs.

6.3.1 Construction of an Axial-Flow fan

The principal components of an axial fan are the impeller, the motor and the casing.

The impeller comprises two parts, the hub and the blades. Blades may be manufactured from flat plate or cast from alloy in an aerodynamic shape. It is also possible that the blades are fixed or adjustable.

The number of blades on an impeller may be expressed in terms of 'solidity' with 100% solidity the maximum number of blades for that particular design. Generally speaking the higher the solidity the higher the pressure.

The hub to blade ratio is indicative of the performance of the fan. For a given overall diameter, long blades and a small diameter hub indicates high airflow and low pressure, obviously shorter blades and a larger hub indicates higher pressure and lower airflow.

The clearance between the blade tips and the casing must be kept small to prevent pressure loss. This calls for accurate machining of the casings and blades, resulting in increased cost.

The air leaving the axial-flow fan usually has a rotary motion, or "swirl". This results in a loss of energy. This loss is reduced by the installation of vanes called guide vanes fitted to straighten the flow of air.

Axial-flow fans are particularly suitable for mounting directly in ducts.

A second impeller, rotating in a direction opposite to the first, can also be fitted. This results in the swirl induced by the first set of blades being neutralised by the swirl imparted by the second set. Fans fitted with this type of additional impeller are known as contra-rotating fans

6.4 Centrifugal or Radial-Flow fans



When a stone attached to a string is swung in circles and then let go it will fly outwards. In fact anything that revolves tends to move away from the centre of rotation with centrifugal force. The word centrifugal meaning *"moving or directed away from the centre"*.

The impeller of a centrifugal fan consists of a hollow "wheel", "rotor" or "runner", with a number of blades

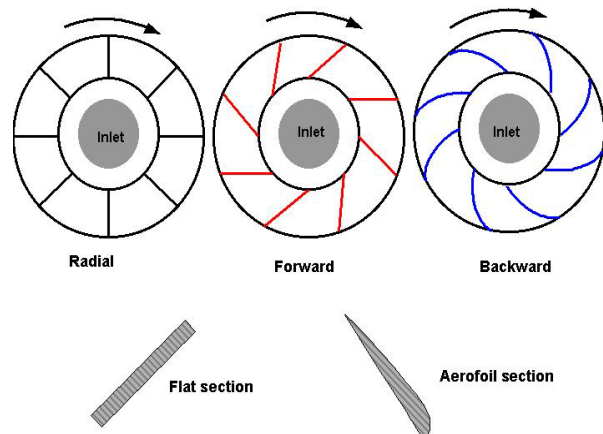


around its circumference, similar to the paddle wheel of old steamships.

The blades are generally relatively short, and can be arranged in three ways:

- Radially
- Forward-curved, or forward-sloping
- Backward-curved, or backward sloping.

When the impeller rotates, the air between the blades is thrown outwards by centrifugal force and leaves the impeller at right angles to the axis. The air is guided by the shape of the fan casing to the fan outlet, and is then replaced by air that enters along the direction of the axis from either one side only, when the fan is termed a “single-inlet” fan, or from both sides, when the fan is termed a “double-inlet” fan.



6.4.1 Construction of a Centrifugal Fan

In practice the shapes of the blades differ considerably in different types of fans. The performance of the fan depends upon the design of the blades and the design of the volute (scroll) which is the section of the fan casing along which the air flows after leaving the impeller. It is the design of the volute that determines the amount of velocity pressure in the air after leaving the blades that will be converted into static pressure.

On some centrifugal fans, a certain amount of control of performance is provided by the installation of variable inlet guide vanes.

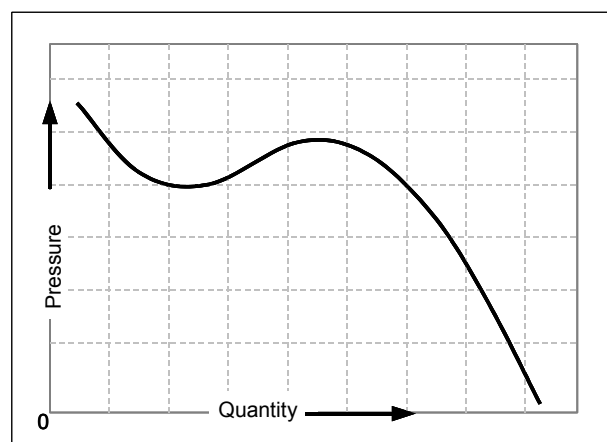
6.5 The Fan Characteristic (Performance) Curve

The performance of a fan is generally presented in the form of a graph on which is plotted the airflow quantity (Q) along the horizontal axis and the fan pressure (P) along the vertical axis. These points are obtained from tests where the actual fan performance is measured. A line joins these points in the form of a curve. This curve is termed the fan characteristic curve. The power and fan efficiency is shown for each airflow point and similarly joined to form a curve.

Fan performance curves are supplied by fan manufacturers and show the predicted airflow volumes for a given pressure.

As the pressure requirements increase, the airflow volume decreases. It should be noted that these curves apply only for the particular fan and at a particular density of air. Any variations of density along with blade pitch angle will result in a change to the characteristic. These changes can be predicted and incorporated into the design. However, there are changes that cannot be predicted and are caused by dirty or worn fan blades.

Fan performance curves are presented in many differing forms. Some examples of fan curves are shown in the following figures.

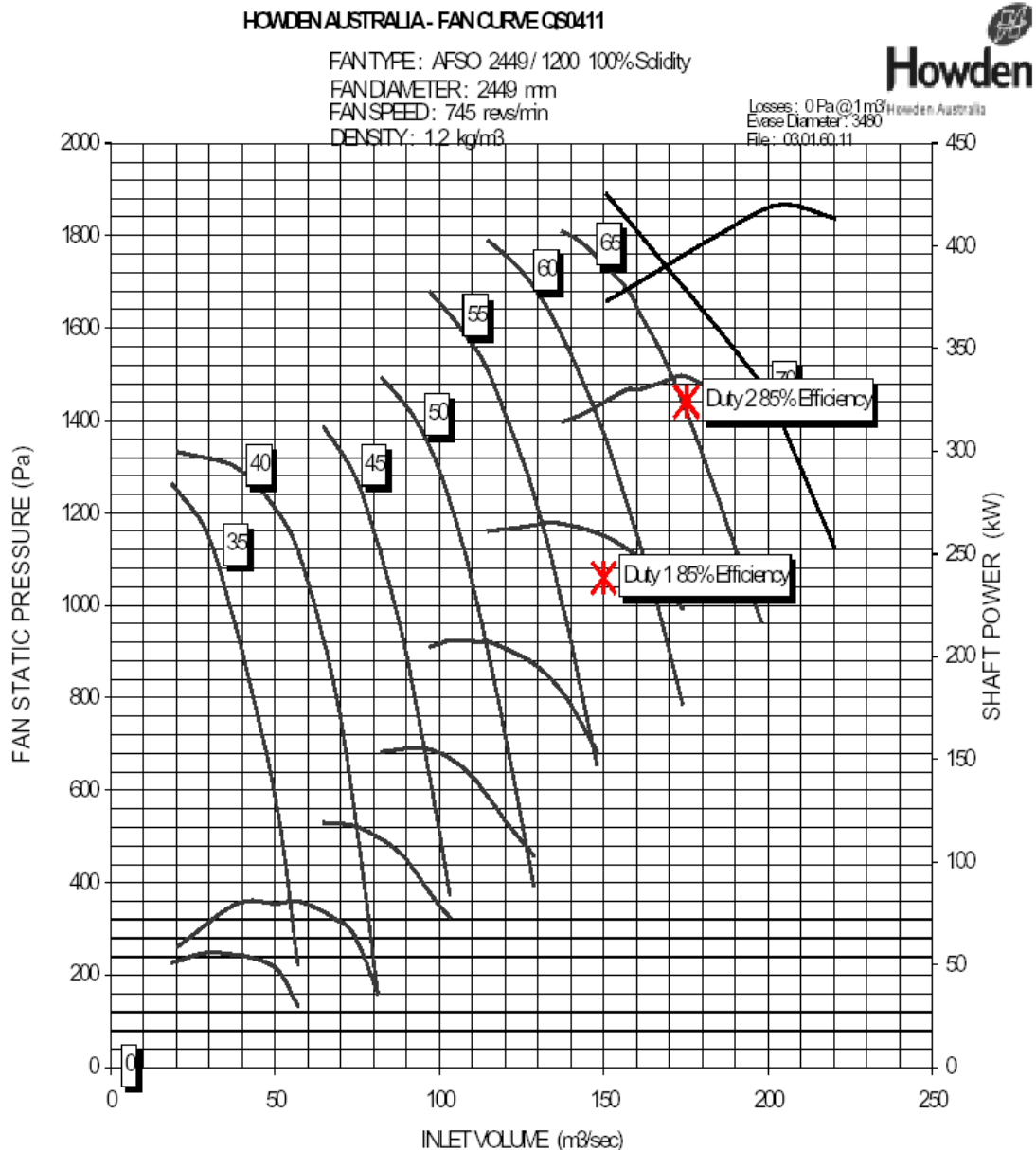


In order to make the correct selection people interpreting these curves should be aware of the following:

1. Two curves represent the total pressure performance whilst the other show static pressure performance.
2. All curves show the fan shaft power expressed in different ways
3. Although only one of the curves shows the efficiency (for the pressure) drawn on the chart it is easily interpreted by calculation from the information provided. (this is discussed later in Section 6.13 and Section 6.14)

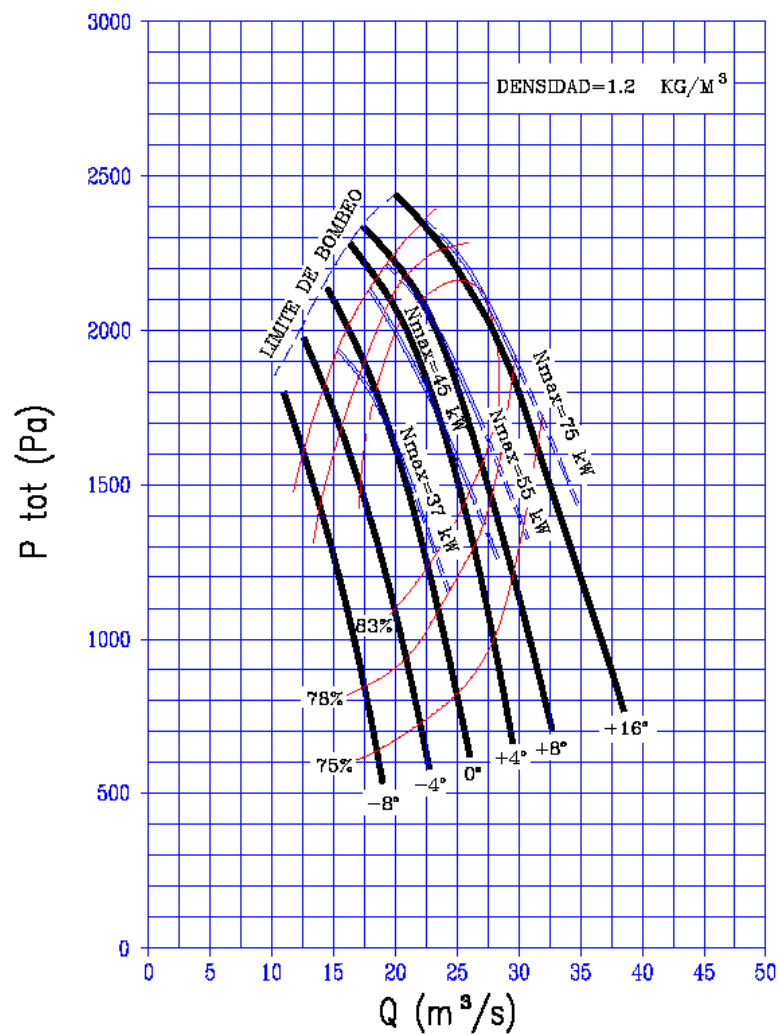
Some examples of fan curves are provided below

SECTION 5 – FAN PERFORMANCE CURVE





VENTILADORES TIPO ZVN 1-12... AXIAL-FANS TYPE ZVN 1-12... AXIALVENTILATOREN
TYPENREIHE ZVN 1-12...



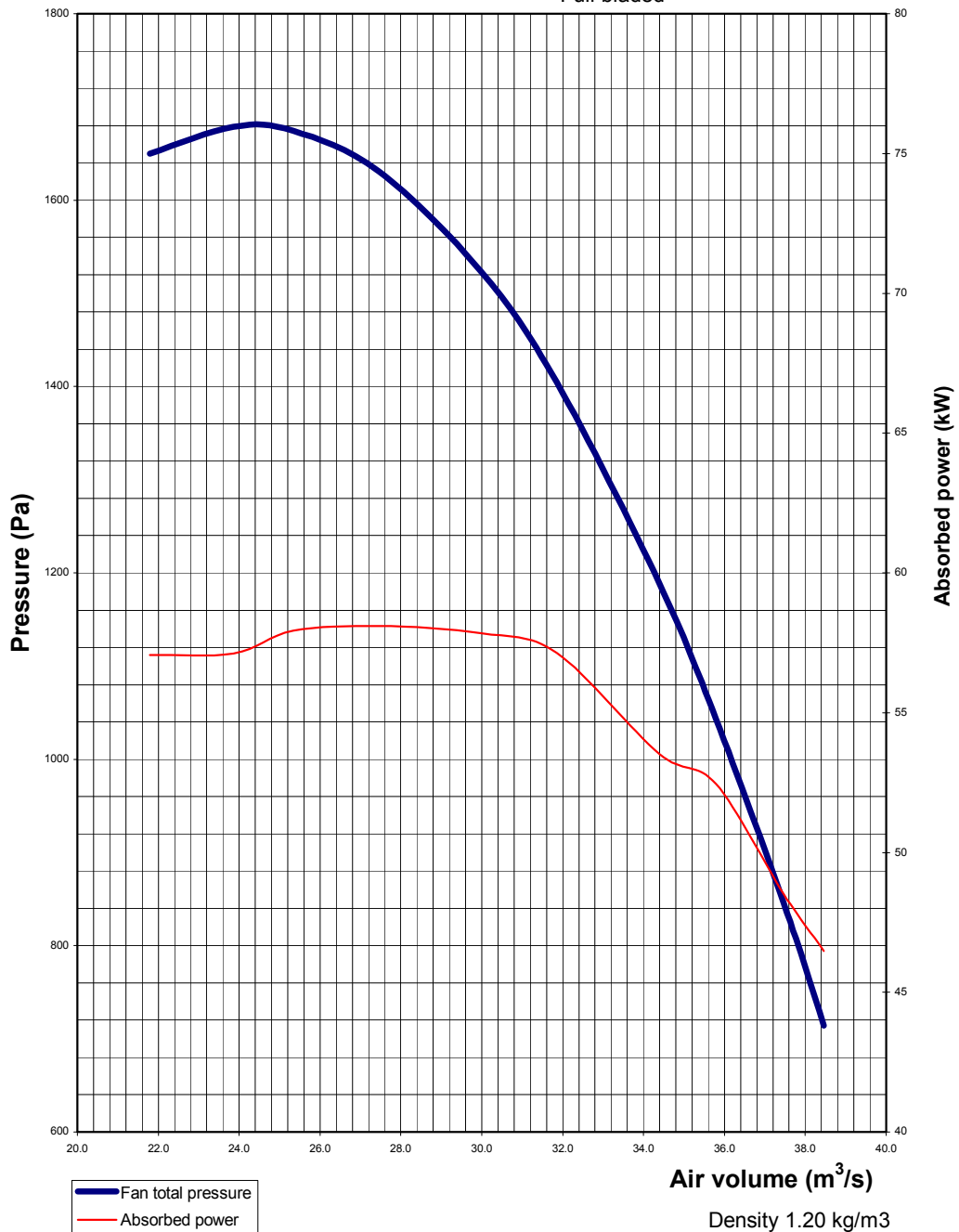
ZVN 1-12-37/4	N=37 kW	n=1500 min ⁻¹
ZVN 1-12-45/4	N=45 kW	n=1500 min ⁻¹
ZVN 1-12-55/4	N=55 kW	n=1500 min ⁻¹
ZVN 1-12-75/4	N=75 kW	n=1500 min ⁻¹

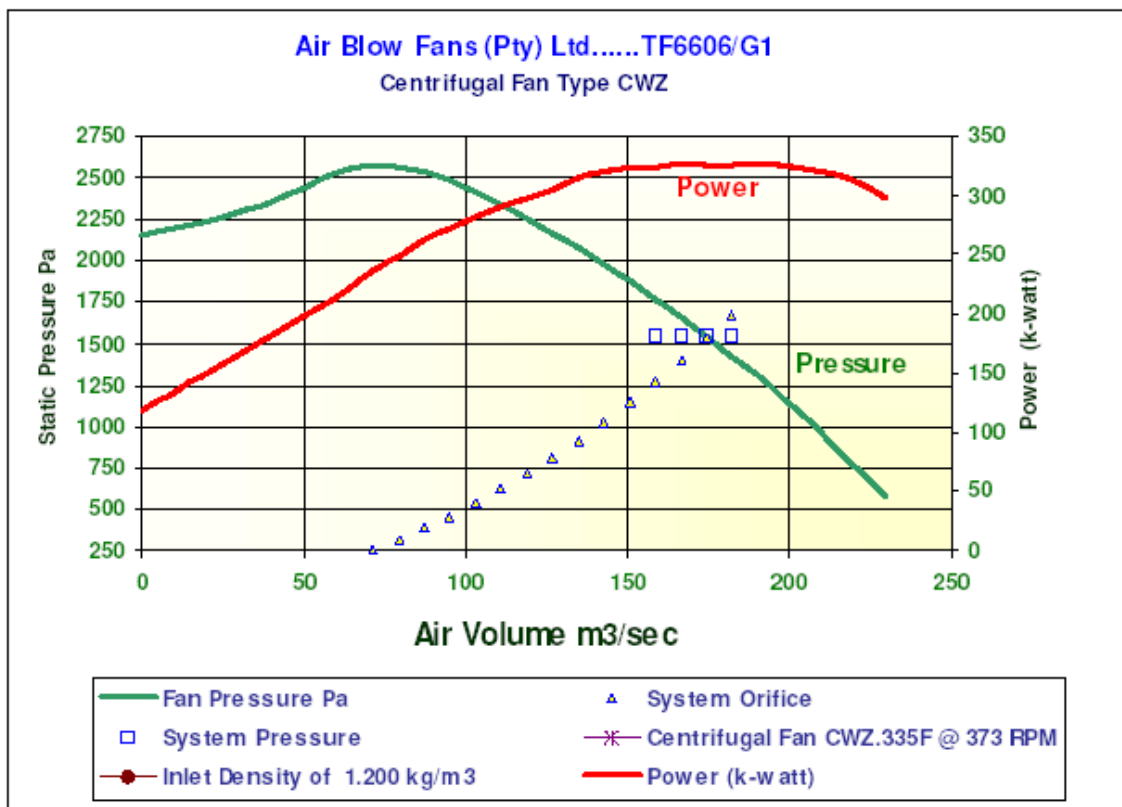
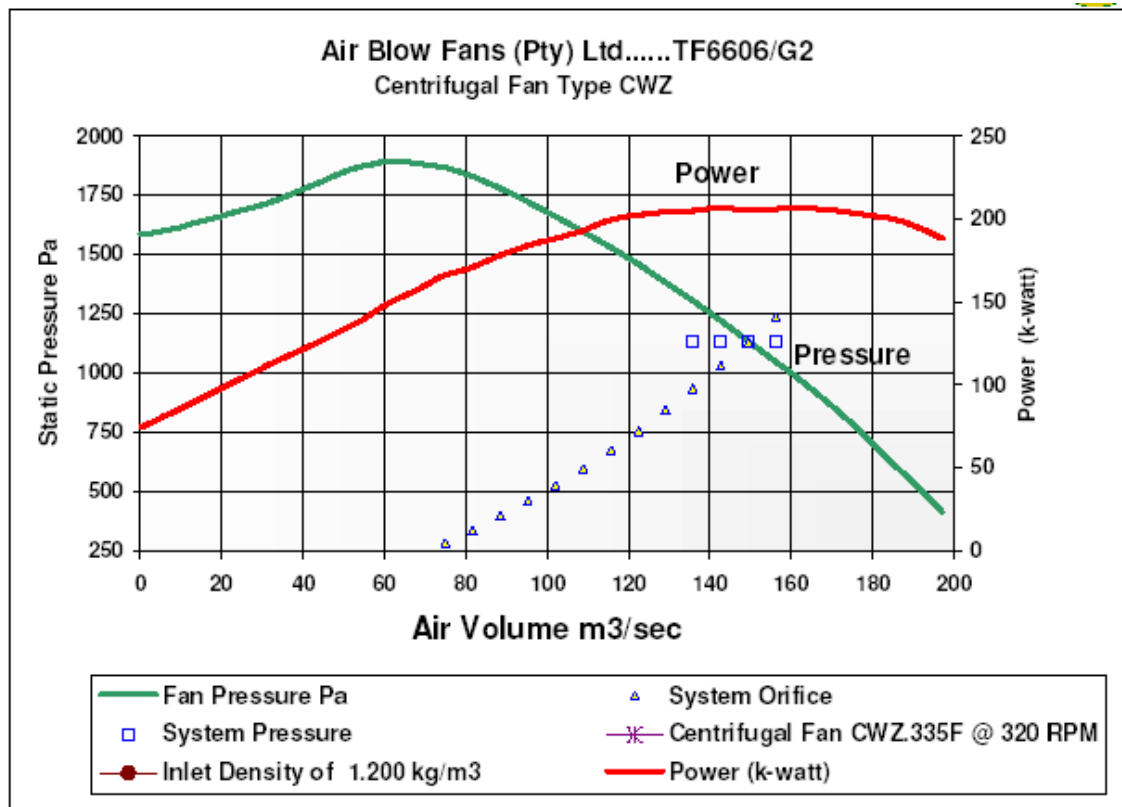


SDS AUSMINCO
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Fx: (08) 9455 1819

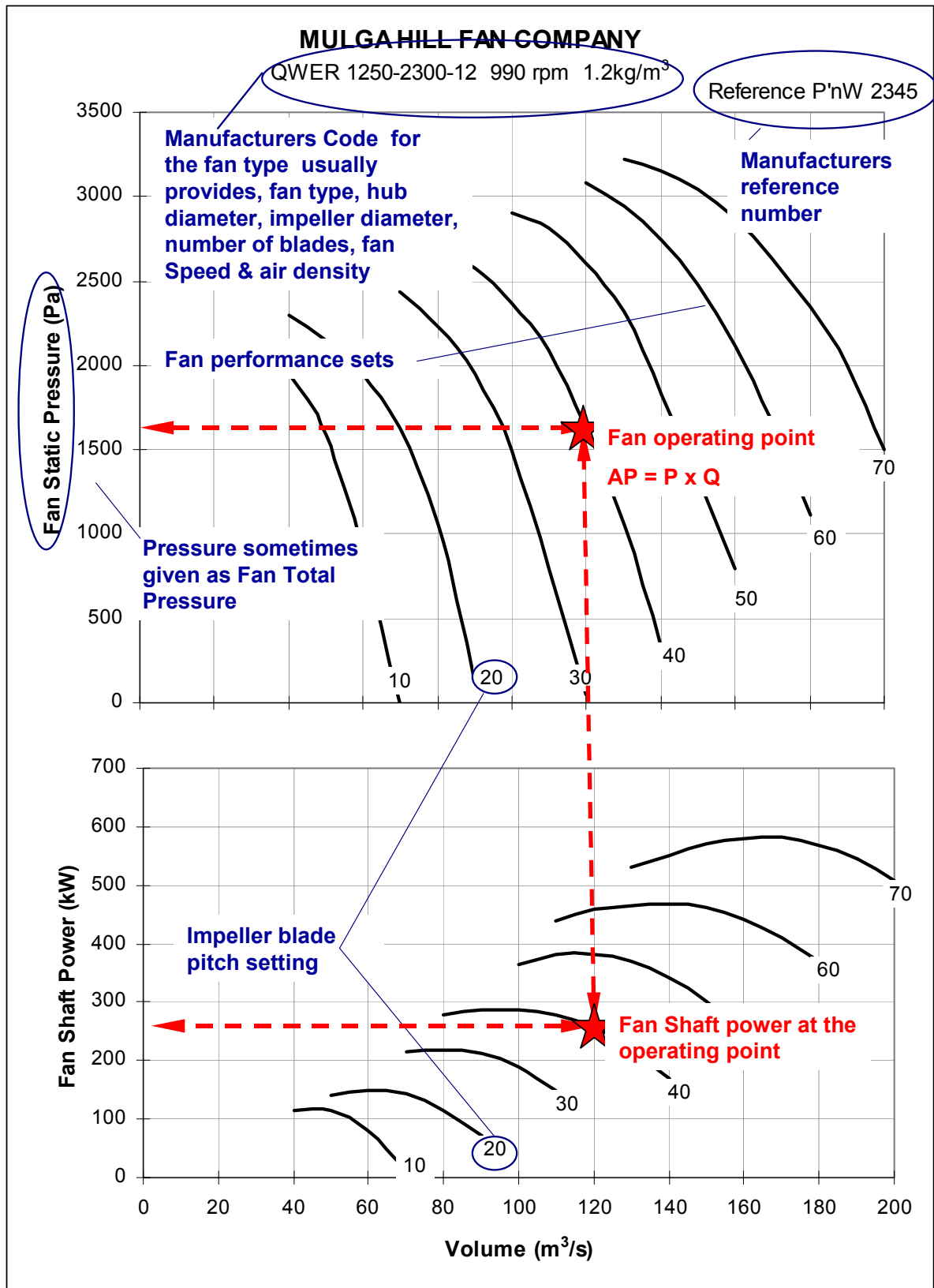
AL12-500 x 1 fan
Diameter: 1200 mm
Speed: 1000 rpm (nominal)
Power: 55 kW
Single stage Axial
Full bladed





6.6 Interpreting Fan Performance Curves

The correct interpretation of fan curves is essential to ensure that the fan selection will meet the requirements.

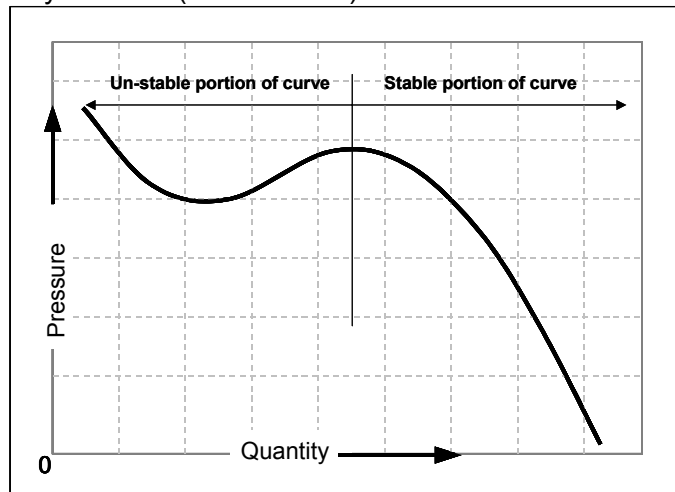


The example curve shown above represents an adjustable pitch axial flow fan. Some points to consider are:

1. A fan designed and manufactured to operate at setting 40 does not mean that it can operate at settings above this point. In this case the fan would typically be fitted with a 300 kW electric motor and if the fan pitch angle were increased the potential for the fan to stall would be extremely high. It would be necessary to increase the size of the motor.
2. It may not be possible to increase the size of the motor because it must be small enough to fit inside the housing and retain the same centreline.

6.7 Fan Stall

A fan will “**stall**” if the resistance of the system is (or becomes) excessive and as this resistance curve becomes steeper the intersection point with the fan performance characteristic changes and moves to a point of higher pressure (lower quantity) on the fan characteristic. At this point, both the fan pressure and airflow quantity increase simultaneously (i.e. both the fan characteristic and the system characteristic have a positive slope) this is opposed to the normal operation where the pressure increases as the quantity decreases. Eventually, the pressure and quantity become unstable and fluctuate wildly. This condition is known as “**hunting**” or “**stall**”.



This zone is usually not shown on manufacturers curves, but it can be assumed to occur above the highest pressure shown. If a fan is operating in this stall zone, the air tends to separate from the blade surfaces, causing excessive vibration. These vibrations may damage the fan if it is allowed to operate for any length of time. This zone may be very noticeable in some fans and hardly noticeable in others.

Stall Zone Signs

1. The fan vibrates excessively
2. The fan makes unusual noises (whines because of increased pressure)
3. Input power amps show rapid oscillations or surging at a slower frequency (every couple of seconds)
4. The quantity and pressure points alter considerably

Action to be taken

Continued operation of a fan in stall may eventually cause damage to the fan and the blades may disintegrate. To remove the fan from this condition some air may be allowed to enter the system at a position close to the fan (ie short circuit the system) this will have the effect of reducing the overall system resistance). This should only be a temporary solution until the cause of the stall is identified and rectified.

6.8 Fan Performance Control

Although fans are required for a particular duty, it should not be forgotten that mining by its very nature is dynamic. Over the life of the mine, fans may be required to operate at a number of

specific duties. To cater for these changes, fans must have flexible characteristics. Some of the causes for differing fan duties include;

- Increased resistance
- increased airflow requirements
- change in air density
- power saving

The output of a fan can be controlled by adjusting the system (mine resistance) or using outlet dampers, inlet-box dampers, variable inlet vanes, variable pitch, variable speed or varying the number of fans.

6.8.1 System Damper

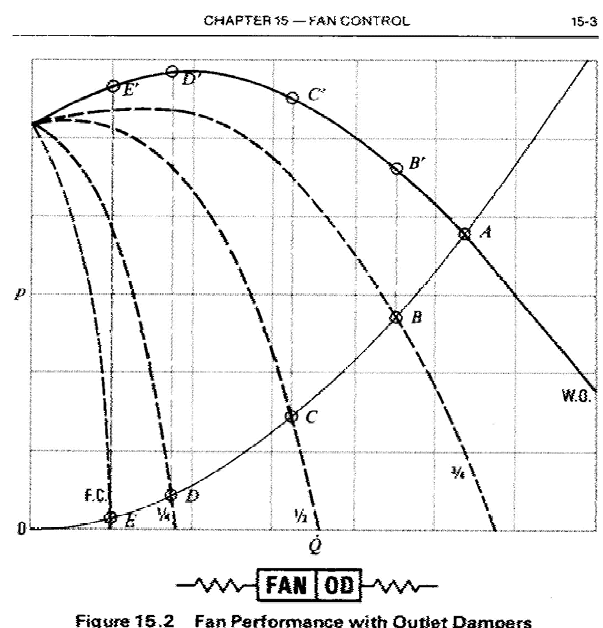
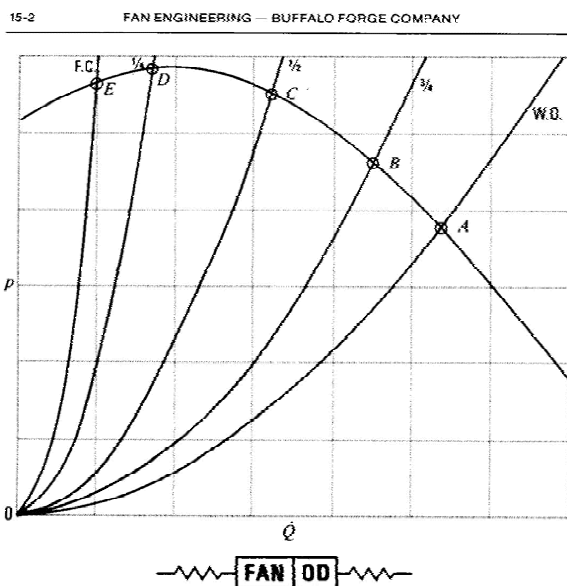
Dampers (regulators) installed in fan ducts (mine openings) to act as throttling devices have the effect of altering the system resistance. The fan will operate at different points on its characteristic without change to the pressure, quantity and power curves.

This method is sometimes used to cause a fan to operate at a more efficient section of its performance characteristic. In these cases efficiency may be improved but, at the cost of decreased airflow and some times increased power. There is usually very little economic justification for the change and is not recommended as a control system.

6.8.2 Outlet Damper

A fan outlet damper acts on the fan in the same way as a system damper i.e. it will change the overall system characteristic by altering the resistance.

Consider the graph to the left in the figure below. Points A, B, C, D & E lie on the intersection of the fan curve and a number of parabolic system characteristics with the same origin. These curves represent the combined system and damper at various settings of the outlet damper control. Point A would represent the damper wide open and the other points represent the damper at other settings up to point E fully closed (FC). In this case the operating points D and E are potentially unstable and it would not be recommended to operate the fan with the damper set in this position.



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It is possible to represent the various settings of the damper as fan characteristic curves similar to that shown to the right in the figure below. Obviously there are an infinite number of settings between fully open and fully closed.

There is a caution when interpreting this graph. Consider the point C ($\frac{1}{2}$ open). In this case the vertical distance from the base line to C is the pressure loss in the system and the vertical distance C-C' is the pressure loss caused by the outlet damper.

There is some suggestion that because the intersection points A, B, C, D, & E all lie on the negative section of the performance characteristics then the operation will be stable even when fully closed. There is some evidence that outlet dampers can be evaluated in this way and IF the principle resistance is on the inlet side of the fan but it is not suitable if the principle resistance is on the outlet side of the fan.

In essence the damper is considered to be part of the system rather than part of the fan.

6.8.3 Inlet-Box Damper

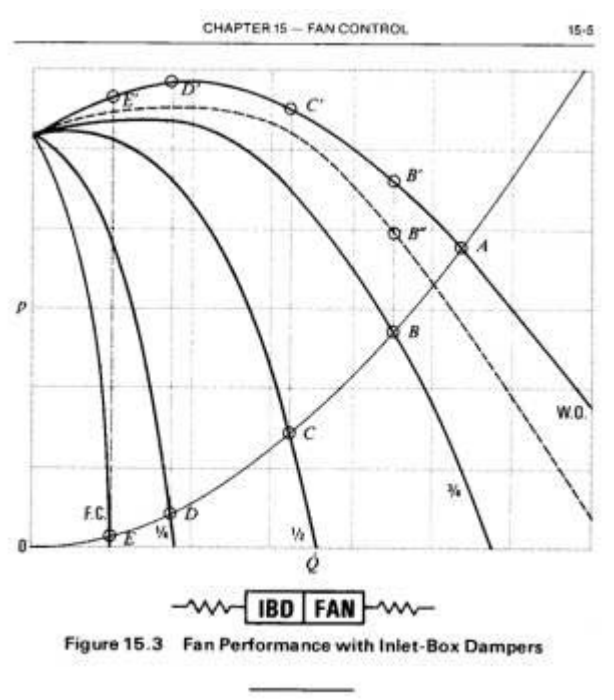
The performance of an inlet-box damper (IBD) is considered in two parts.

The first part is the pressure loss cause by the resistance of the damper, similar to the outlet damper. The second part is due to changes caused by the inlet-box damper to the swirl of the air onto the impeller. This pre-swirl reduces the amount of work for the impeller therefore affecting the impeller output and resulting in a different performance characteristic.

The performance of an inlet-box damper can be considered in the manner shown in the figure opposite.

Here again there is some evidence to suggest that IF significant swirl exists then the intersection points A, B, C, D, & E all lie on the negative section of the performance characteristics and the operation will be stable. There is also some evidence that if the principle resistance is on the outlet side even without pre-swirl the operation will be stable.

However, if the principle resistance is on the inlet side then the operating points should be considered as A', B', C', D', & E' and in this case the points D' & E' lie on the positive part of the fan characteristic and therefore the operation is likely to be unstable. Like the outlet damper the inlet-box damper is generally considered to be part of the system rather than part of the fan.



6.8.4 Variable Inlet Vanes

Variable inlet vanes (VIV's) are similar in appearance to a camera iris and is slightly superior to inlet damper control in respect of efficiency and stable operation.

VIV's provide the same effect on fan performance as the inlet-box damper because they are designed to create a pre-swirl to the air. Again they have two components that caused by the resistance of the structure and that caused by the pre-swirl given to the air.

VIV's are always installed as part of the fan and very close to the impeller. Because of this operating points are always considered to be points A, B, C, D, & E and highly unlikely to operate in the unstable positive characteristic points D' & E'.

The use of VIV's in Australian mines is usually confined to primary centrifugal fans but their use with axial flow fans is common particularly in the deep mines of southern Africa.

Maintenance of this component is usually very poor and most become seized at a particular setting. This component should be greased weekly and exercised daily. Exercising requires the vanes to be closed then again open to the desired setting. This is a very simple operation that can be undertaken automatically, particularly given today's technology of PLC's.

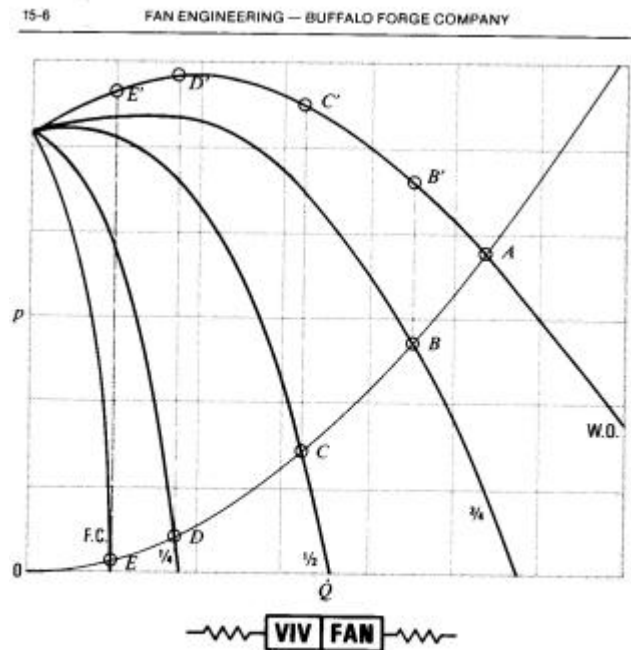


Figure 15.4 Fan Performance with Variable Inlet Vanes

6.8.5 Variable Speed

Speed control can be achieved in a number of ways

- Changing drive pulley ratios.
- Reduction gearbox drives.
- Variable speed controllers
- Number of poles in the motor

$$\text{Motor Speed} = \frac{120 \times \text{Hz}}{\text{No of Poles}} \quad (\text{can only be an even number})$$

Variable speed characteristics can easily be determined from the fan laws detailed in section 6.9. The performance characteristic curves would be identical except that all devices to alter the speed must have some slip to transmit torque.

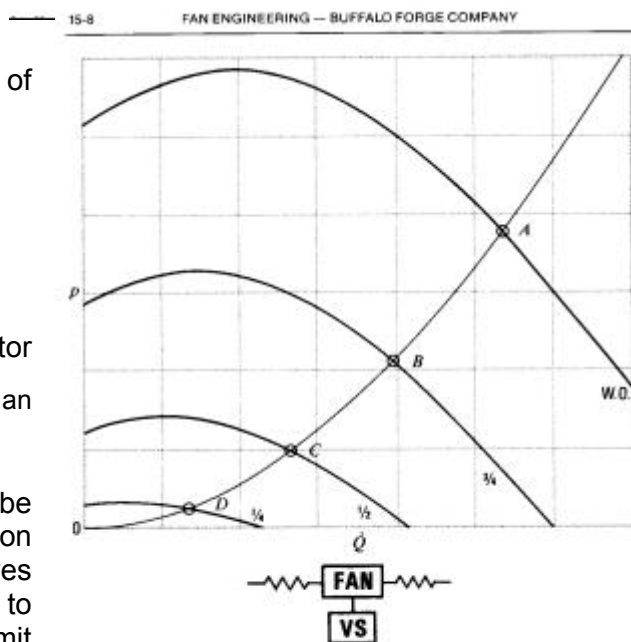


Figure 15.6 Fan Performance with Variable Speed

6.8.6 Variable Pitch

The figure opposite shows the performance characteristic for a fan and the operation points with the blades set at different angles. The system is depicted by the curve drawn through points E, D, C, B, A & Z. A fan would normally be selected to operate a point A as this would be the most

efficient use of power. Higher flow rates would be achieved at point Z and lower rates at points B, C, D, & E.

Altering the blade pitch is normally done by stopping the fan, loosening the nut holding the blade to the hub and then turning the blade to the required pitch setting. The nut is then again tightened to secure the blade in its new position. On certain fans, this adjustment can be carried out by means of a single lever or series of linkages when the fan is in operation. These are termed variable-pitch in motion fans.

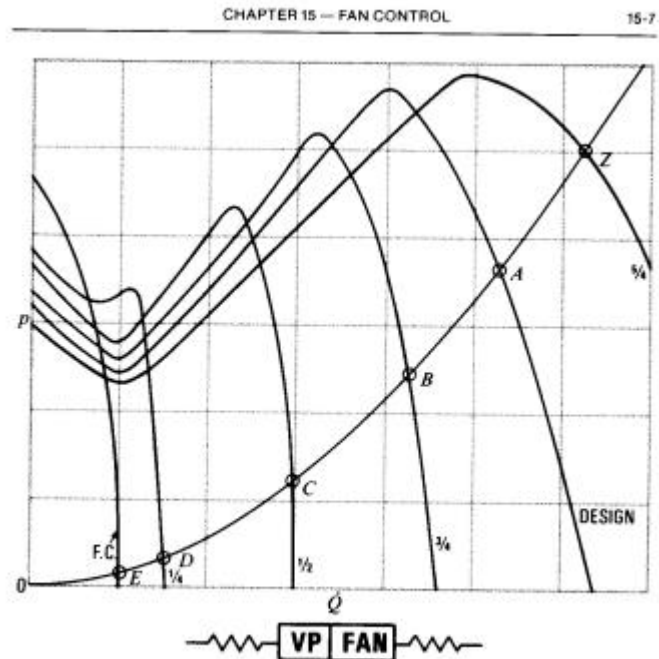


Figure 15.5 Fan Performance with Variable Pitch

6.8.7 Fans in Series

If we assume that air is incompressible then the potential performance of two identical fans is obtained simply by multiplying by two the pressure at a particular quantity.

In the figure opposite one fan will operate in the system at B by adding an identical fan the combined operating point will be at A.

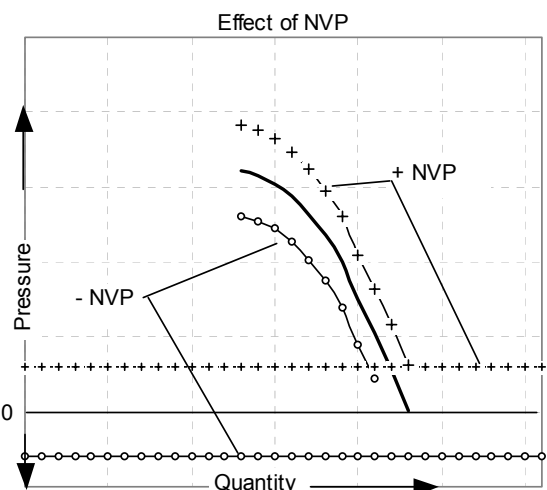
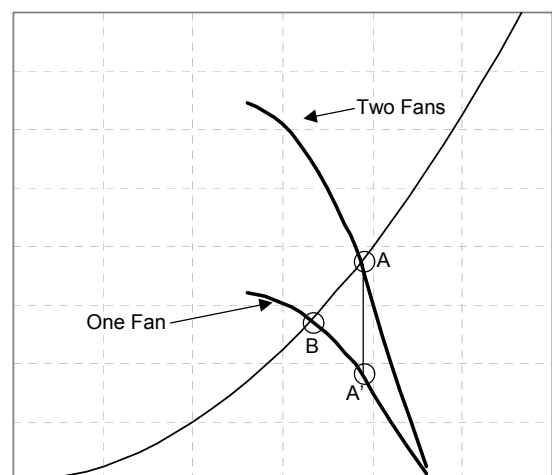
Fans in series are generally well recognised as most auxiliary development fans are configured to operate in this manner and there are many instances where primary ventilation fans have two stages.

Fans are generally installed in series to overcome an increased system resistance or maintain a particular quantity.

In most cases fans installed in series are designed for that purpose. However it is possible to install unlike fans in series but this is the exception rather than the rule.

NOTE that the NVP acts in the same way as a series fan with the exception that NVP can have a negative pressure that is subtracted from the fan pressure characteristic.

When two unlike fans are installed in series then the combined pressure generated at a given airflow quantity will be the sum of the pressures of the individual fans at that quantity. The



exception being contra rotating fans when the pressure can be three times that of each individual fan.

In practice, the overall flow is increased with the installation of a second fan in series however, in most auxiliary fan duct installations there will be little to no increase in flow discharged at the end of the duct, only an increase in leakage along the length of the duct.



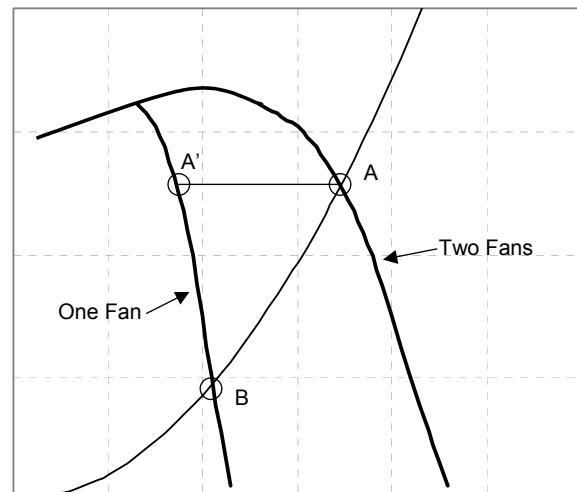
6.8.8 Fans in Parallel

The potential performance of two identical fans operating in parallel is obtained simply by multiplying by two the quantity at a particular pressure.

In the figure opposite one fan will operate in the system at B by adding an identical fan the combined operating point will be at A.

When fans are selected to operate in parallel, it is essential to ensure they operate in the stable area of the combined characteristic.

When combining parallel fan performance



curves the only the quantities are added and the pressure will remain the same.

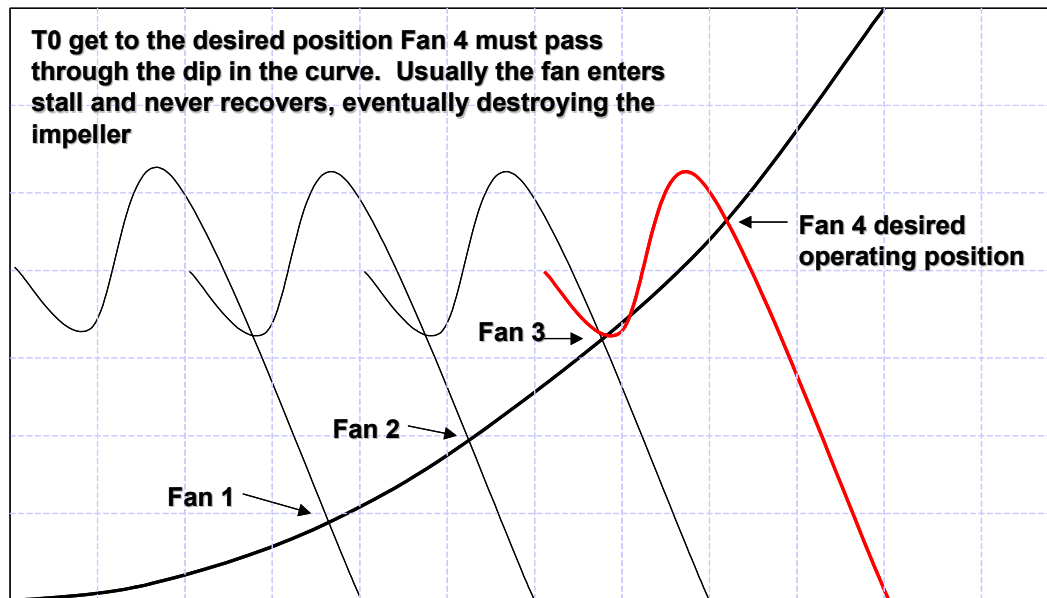
The important thing is to note is that both fans **MUST** operate at the same pressure.

When unlike fans are installed in parallel they will provide different quantities of air but will operate at the same pressure.

The use of fans in parallel is not uncommon in mines and in fact many mine circulating large quantities use fans in parallel as it offers the opportunity for maintaining some airflow should one fan fail. Some mine choose to install two or even three fans and use one as an insitu spare and only operated in an emergency when an operating fan fails.

The use of multiple fans in parallel may cause problems when starting. Some fan performance curves have a decided dip in the performance and if the fan being started has to pass through this dip or the positive slope side of the performance characteristic then it will enter the “Stall” position and never recover.

Fan manufacturers seldom show the positive slope side of the curve and it can be misleading. As a precaution the fan manufacturer must be questioned specifically as to the possibility of having to pass through the dip on the stall side of the fan performance.



6.8.9 The ‘Eck’ Line

When a fan enters the unstable part of its curve it will start to stall. Operating two or more fans in parallel, will induce stalling at a lower operating pressure. This is because succeeding fans must start up against the pressure already being developed by the operating fan or fans.

The calculated unstable portion of the combined curve is known as the ‘Eck’ line¹⁷. The determination of this line will show the lower “maximum useful pressure” to ensure operation in the stable portion of the combined fan performance curve.

Consider the combined curves for one, two and three identical fans in parallel shown in the figure below. The combined ‘Eck’ lines can be determined in the following manner.

For a **single fan** the unstable portion of the performance curve is shown as a dashed line and the fan can be operated at any point to the right of the dashed line.

For **two fans**, the combined curve (labelled 2 Fans) shows two unstable portions.

The first is due to the combination of the two fan curves and is shown as a bold black line. Point ‘A’ on this curve is obtained by doubling the air quantity at point ‘a’ on the single fan curve.

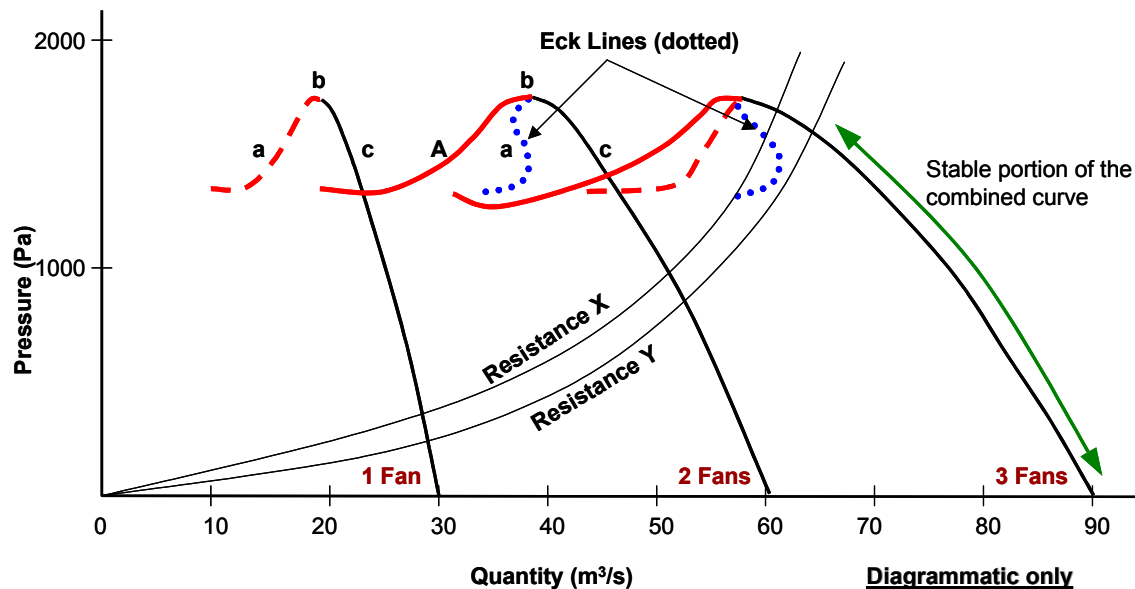
The second unstable portion of the two-fan system is due to the unstable part of each fan and is shown as a dotted line. This portion is constructed by marking a point to the left of the combined curve at distance ‘a-c’ as measured for a single fan. Other points on this unstable portion of the curve are constructed in the same manner to obtain the dotted line for two parallel fans. Hence for two fans in parallel, the stable portion of the fans is to the right of the curve ‘a-b’.

Finally, the curve for **three fans** in parallel (labelled 3 Fans) has three unstable portions:

¹⁷ ‘Eck’ - In recognition of the work undertaken by Professor Bruno Eck on this subject.

1. due to the combined P-Q curve for three fans;
2. due to the unstable portion of two fans; and
3. due to the unstable portion of a single fan.

The stable part of the curve lies to the right of the 'Eck' line.



In the figure above, two system resistance curves are plotted. Resistance X has a resistance higher than Resistance Y.

If these fans were operated in parallel against Resistance X, the third fan would have to pass through the unstable part of the combined curve on the run up to the system pressure. In reality, this would not occur and the fan would stall.

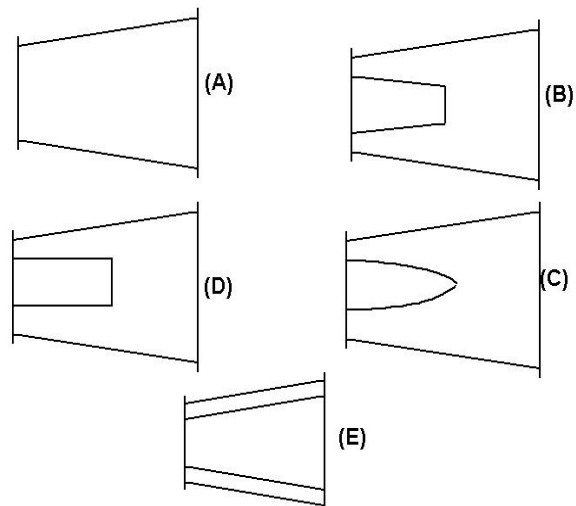
If these fans were operated in parallel against Resistance Y, the third fan would run up to the system pressure by passing underneath the 'Eck' line. In this case the probable maximum duty would be 65 m³/s at 1,500 Pa.

6.8.10 Fan Diffuser

Fan diffusers ultimately cause a little extra airflow through the fan. As shown in Section 2 system shock losses can be reduced by the installation of an effective diffuser at the discharge.

The diffuser may be fitted with an internal fairing. They vary in shape from rectangular to round and the included angle, the outlet to inlet ratio and the cross sectional shape affect the efficiency.

The best pressure regain is achieved on a diffuser without an inlet fairing, and with an outlet to inlet area ratio of 4:1 and the optimum included angle between 8° and



11°. Because of the length required for this optimum in practice this angle can be much larger (up to 25°)

$$P_{V \text{ REGAINED}} = \eta (P_{V \text{ INLET}} - P_{V \text{ OUTLET}})$$

Where

$$\eta = \text{Efficiency of the evasè (\%)}$$

6.9 Fan Laws

Fan performance curves are determined for a specific speed and density.

Unless otherwise specified the density for a fan curve can be assumed as standard air (1.2 kg/m³). Standard air is a definition of convenience that is widely adopted. The actual values of temperature, pressure and moisture content are not important.

Should the speed or density of the fan inlet air be altered, the following fan laws will apply.

Change	Density (ρ)	Speed (rpm)	Diameter
Quantity (Q)	$Q_1 = Q_2$	$\frac{Q_1}{Q_2} = \frac{\text{rpm}_1}{\text{rpm}_2}$	$\frac{Q_1}{Q_2} = \left(\frac{D_1}{D_2}\right)^3$
Pressure (P)	$\frac{P_1}{P_2} = \frac{\rho_1}{\rho_2}$	$\frac{P_1}{P_2} = \left(\frac{\text{rpm}_1}{\text{rpm}_2}\right)^2$	$\frac{P_1}{P_2} = \left(\frac{D_1}{D_2}\right)^2$
Power (kW)	$\frac{\text{kW}_1}{\text{kW}_2} = \frac{\rho_1}{\rho_2}$	$\frac{\text{kW}_1}{\text{kW}_2} = \left(\frac{\text{rpm}_1}{\text{rpm}_2}\right)^3$	$\frac{\text{kW}_1}{\text{kW}_2} = \left(\frac{D_1}{D_2}\right)^5$
Efficiency (%)	$\eta_1 = \eta_2$	$\eta_1 = \eta_2$	$\eta_1 = \eta_2$

When considering the diameter the fans must have the same geometric dimensions, and if the size difference is great then it can be expected that the fan total pressure may be larger than that given by these laws (scale effect).

It is also possible that a change in density due to compression of the air will occur affecting the results from the use of these laws. This is normally only of concern at pressures above 2.5 kPa.

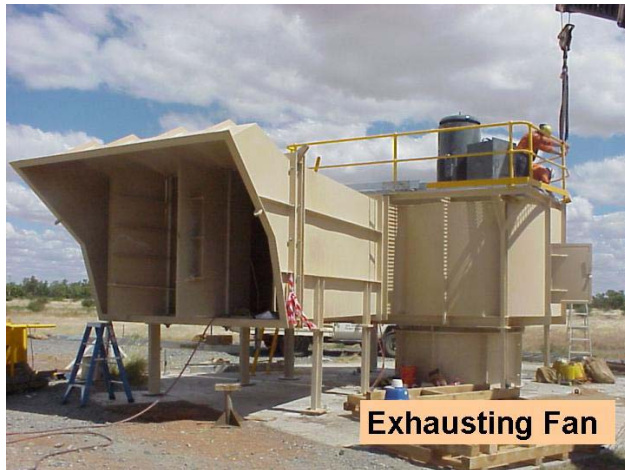
This is discussed more fully in “Woods Practical Guide to Fan Engineering” or “Fan Engineering” published by the Buffalo Forge Company.



6.10 Measuring Fan Performance

Performance testing is undertaken to determine the actual operating performance of fans particularly at time of installation for comparison with the manufacturer's predicted performance.

Fan performance testing is detailed in standards such as Australian Standard 2936 and British Standard 848 and is usually specified in fan tender documents as the measure of compliance.



Standards tests are performed using standardised airways with devices fitted to straighten flow and alter the operating pressure. The installation of such components for site measurements is generally considered to be too costly and is seldom (if ever) used.

Precision laboratory type testing cannot be carried out on site. Measurements made to determine site performance must be agreed with the fan manufacturer at the time of the purchase order being placed.

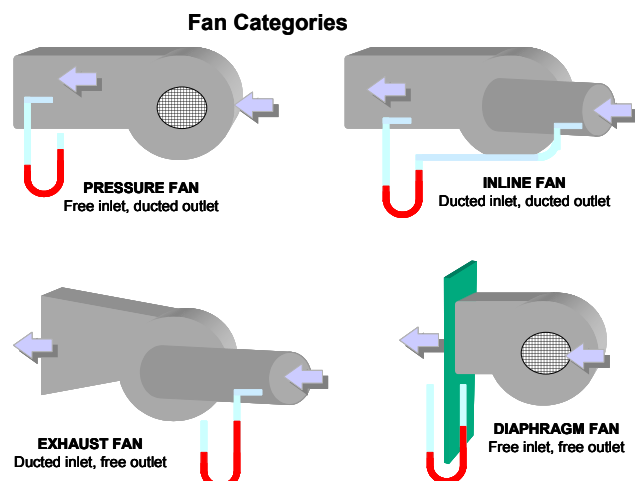
Aside from the regular three monthly intervals to comply with some mining legislation the

measurement of fan performance is also undertaken:

1. For acceptance testing at time of purchase and commissioning.
2. Prior to making an operational change to the fan (i.e. altering the speed, changing a motor etc.).
3. After making an operational change to the fan to ensure the desired (predicted) result has been achieved.
4. To determine if a measured decrease in system airflow quantity is due to system resistance or the operational performance of the fan.

The main objective of measurements is to accurately determine:

1. fan inlet airflow quantity
2. fan pressure
3. fan efficiency



Measurements of fan performance are made to represent the actual airflow through the fan and must therefore be sited to ensure there is no change in the properties of the airflow between the measuring site and the fan inlet.

Such changes may include density, caused by differing temperature and pressure as well as airflow quantity caused by leakage into or out of the system.

Fans are primarily categorised as pressure, exhausting, in line, or diaphragm mounted.

6.10.1 Fan Total Pressure

The fan total pressure (FTP) is equal to the total pressure at the fan outlet (TP_O) minus the total pressure at the fan inlet (TP_I).

$$FTP = TP_O - TP_I \quad \text{Equation 6-1 Fan Total Pressure}$$

6.10.2 Fan Velocity Pressure

The fan velocity pressure (FVP) is equal to the velocity pressure at the fan outlet (VP_O).

$$FVP = VP_0 \quad \text{Equation 6-2 Fan Velocity Pressure}$$

6.10.3 Fan Static Pressure

The fan static pressure (FSP) is equal to the FTP minus the fan velocity pressure.

$$FSP = FTP - FVP \quad \text{Equation 6-3 Fan Static Pressure}$$

6.11 Pressure Fans

The figure below shows the side tube and facing tube measurements for a pressure (forcing) fan (most mine secondary fans). From these measurements of pressure we can determine the Fan Pressures

By definition $FTP = TP_0 - TP_1$

in this case $TP_1 = 0$

therefore $FTP = 2000 - 0$
 $= 2000 \text{ Pa}$

By definition $FVP = VP_0$

From the equation for ventilation pressure $TP = SP + VP$

then $VP_0 = TP_0 - SP_0$
 $= 2000 - 1500$
 $= 500 \text{ Pa}$
 $= FVP$

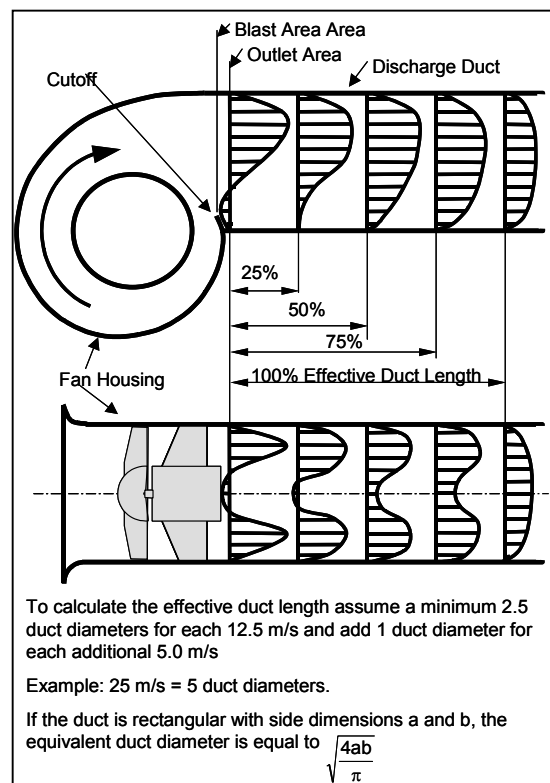
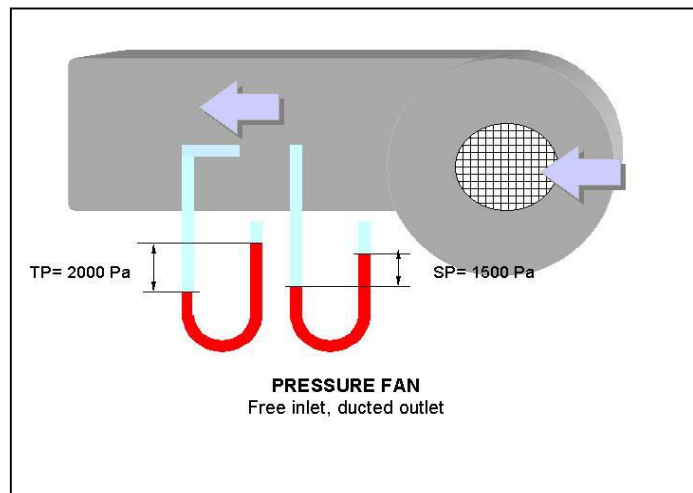
and $FSP = FTP - FVP$
 $= 2000 - 500$
 $= 1500 \text{ Pa}$

Therefore, a static pressure reading downstream of a pressure fan is equal to the Fan Static Pressure (minus the frictional pressure losses between the fan outlet and the measuring site).

6.11.1 Selection of Measuring Site

Satisfactory measurements can only be made in a plane where the airflow is substantially free of swirl and air turbulence and generally should be no closer than 1.5 duct diameters upstream of the fan inlet or 2.5 diameters from the fan outlet.

It is preferred that airflow measurements are made upstream of the fan (the fan intake). If it is necessary to measure downstream (fan discharge) this distance should be equal to the effective duct length (described in the figure opposite). This is to allow for the velocity profile to become evenly distributed in the duct.



6.12 Exhausting Fans

The figure below shows the side tube and facing tube measurements for a suction (exhausting) fan (as installed for most mine primary ventilation). From these measurements we can determine the fan pressures.

In this case, the static pressure at the fan outlet (SP_0) would equal zero.

Since $TP = SP + VP$

And $SP_0 = 0$

Then $TP_0 = VP_0$

By definition $FSP = FTP - FVP$

And by substitution

$$FSP = (TP_0 - TP_1) - VP_0$$

Since $TP_0 = VP_0$

Then by substitution

$$FSP = VP_0 - (TP_1) - VP_0$$

$$= -TP_1$$

$$= -2580 \text{ Pa}$$

Therefore, a facing tube measuring the inlet total pressure of an exhausting fan is equal to the FSP.

Now as $SP_1 = TP_1 - VP_1$

And $TP_0 = VP_0 = VP_1$ (no change in diameter)

then by substitution $SP_1 = TP_1 - TP_0$

Rearranging $SP_1 = TP_0 - TP_1$

and as $TP_0 - TP_1 = FTP$

then $FTP = -SP_1$

$$= -(-2000)$$

$$= 2000 \text{ Pa}$$

Therefore, a side tube measuring the static pressure of an exhausting fan is equal to the FTP (fan total pressure).

Examples

(Qi) The following measurements were recorded at a surface fan installation exhausting air from a mine. Determine the FSP, FVP and FTP.

Total Pressure	1200
Static Pressure	1800

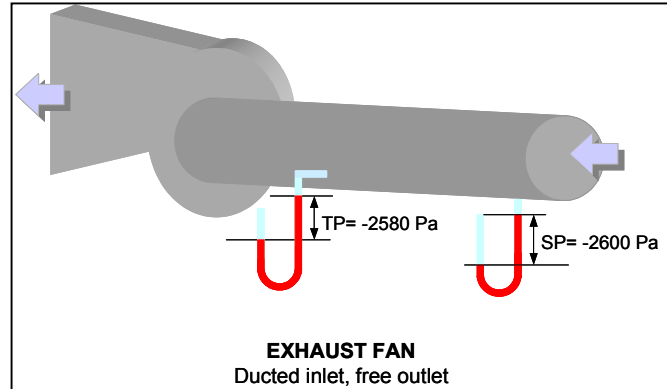
(Ai) Both measurements would be negative ie. on the suction side of a fan.

$$FSP = -1200 \text{ Pa}$$

$$FTP = -1800 \text{ Pa}$$

$$FVP = VP_0 = VP$$

$$= (-1200) - (-1800)$$



$$= -1200 + 1800$$

$$= 600 \text{ Pa}$$

Note Velocity Pressure is always positive.

(Qii) The following measurements were made downstream of a development fan in a 1200 mm diameter duct at a site 4 duct diameters from the fan discharge. No duct was installed upstream of the fan.

Average velocity pressure 480 Pa
Duct Static Pressure 2500 Pa

Determine the FSP.

(Aii) In a pressure system the FSP is equal to the static pressure of the duct.

$$\text{FSP} = \text{SP}_0 = 2500 \text{ Pa}$$

less the frictional pressure losses in the duct.

$$\text{Duct area} = \pi \left(\frac{1.2}{2} \right)^2$$

$$= 1.13 \text{ m}^2$$

$$\text{Duct Perimeter} = \pi 1.2$$

$$= 3.77 \text{ m}$$

$$\text{Assume } K = 0.003$$

$$\text{Then } R = \frac{0.003 \times 3.77 \times (4 \times 1.2)}{(1.13)^3}$$

$$= 0.037 \text{ N s}^2 \text{ m}^{-8}$$

$$\text{Velocity of airflow} = \sqrt{\frac{2VP}{\rho}}$$

$$= \sqrt{\frac{2 \times 480}{1.2}}$$

$$= 28.3 \text{ m/s}$$

$$\text{Airflow} = V \times A$$

$$= 28.3 \times 1.13$$

$$= 32 \text{ m}^3/\text{s}$$

$$\text{and } P = RQ^2$$

$$\text{duct losses} = 0.037 \times (32)^2$$

$$= 37.8$$

$$= \text{Say } 40 \text{ Pa}$$

$$\text{Fan SP} = 2500 - 40$$

$$= 2460 \text{ Pa}$$

6.13 Air Power

Air power is the energy required to move a specific quantity (Q) of air over a specific resistance and can be calculated from

$$\text{AP} = P \times Q \quad \text{Equation 6-4 Air Power}$$

Where AP = Air Power (watts)

P = Pressure (Pa) and also = N/m^2

Q = Quantity of air (m^3/s)

Substituting

$$\begin{aligned} P \times Q &= \text{N/m}^2 \times \text{m}^3/\text{s} \\ &= \text{Nm/s} \end{aligned}$$

and 1 Nm = 1 Joule and 1 Joule/second = 1 watt

therefore AP = watts

Example

A fan requiring 1500Pa to move $400\text{m}^3/\text{s}$ of air has an air power of

$$\begin{aligned} AP &= 1500 \times 400 \\ &= 600,000\text{watts} \end{aligned}$$

6.14 Fan Efficiency

Fan efficiency is important as it determines the cost of power necessary to operate the fan. For example a primary ventilation fan providing $250\text{m}^3/\text{s}$ at a pressure of 2000Pa will absorb 667kW of electrical power when operating at an efficiency of 75% and only 588kW when operating at an efficiency of 85%. Power costs at remote mine sites can be as high as \$0.17/kW hour, therefore over one year the savings are considerable. In this case \$116,800 each year of operation.

The efficiency (η) of the fan is calculated from the equation $\eta = \frac{Q \times P}{W}$. In this equation “W” is the impeller power measured with a swinging frame dynamometer. Because suitable dynamometers are not readily available this is usually measured as the motor input power (MIP). The multiplication of air quantity (Q) and pressure (P) results in the power exerted to the air and is simply termed “Airpower” (AP).

The equation for fan efficiency (η) is therefore usually written as $\eta = \frac{AP}{MIP}$

The efficiency of any fan can be expressed as either Fan Total Efficiency (FTE) where the FTP is used to determine the Airpower, Fan Static Efficiency (FSE) where the FSP is used to determine the Airpower, and the “Fan Overall Efficiency” (FOE). In the case of FOE this includes losses in the electric driving motor, any speed change device between the motor and the impeller and any other source of power.

Caution must be used when interpreting fan curve efficiency. Some curves show static efficiency. Others show total efficiency.

Of particular interest to ventilation practitioners is the overall fan efficiency as this determines the final electrical input power to the motor.

$$\text{FOE} = \frac{\text{Air Power}}{\text{Motor Input Power (MIP)}} \times 100$$

This is often mistaken as the Static or Total efficiencies that do not include the power losses for gearboxes, motors etc.

Example

Determine the FSE and FTE for a mine exhausting fan delivering 300m³/s of air. The fan inlet total pressure has been determined as 1500Pa and the Inlet Static pressure as 2000Pa. The motor input power is 700kW.

For an exhausting fan

$$\begin{aligned}\text{FSP} &= \text{Total Pressure measured at the Fan Inlet} \\ &= 1500\text{Pa}\end{aligned}$$

$$\begin{aligned}\text{Overall FSP efficiency} &= \frac{\text{AP}_{\text{FSP}}}{\text{MIP}} \\ &= \frac{1.500 \times 300}{700} \\ &= \frac{450}{700} \\ &= 64\%\end{aligned}$$

$$\begin{aligned}\text{and FTP} &= \text{Static pressured at the fan outlet relative to atmosphere} \\ &= -(-2000) \\ &= 2000\text{Pa}\end{aligned}$$

$$\begin{aligned}\text{Therefore the Total Efficiency} &= \frac{\text{AP}_{\text{FTP}}}{\text{MIP}} \\ &= \frac{2.000 \times 300}{700} \\ &= \frac{600}{700} \\ &= 86\%\end{aligned}$$

6.15 Measurement of Airflow

The most common methods used to determine airflow in ducts is with an anemometer or a pitot-static tube. Anemometers are used most often in walk-in airways with velocities between 0.2m/s and 14m/s.

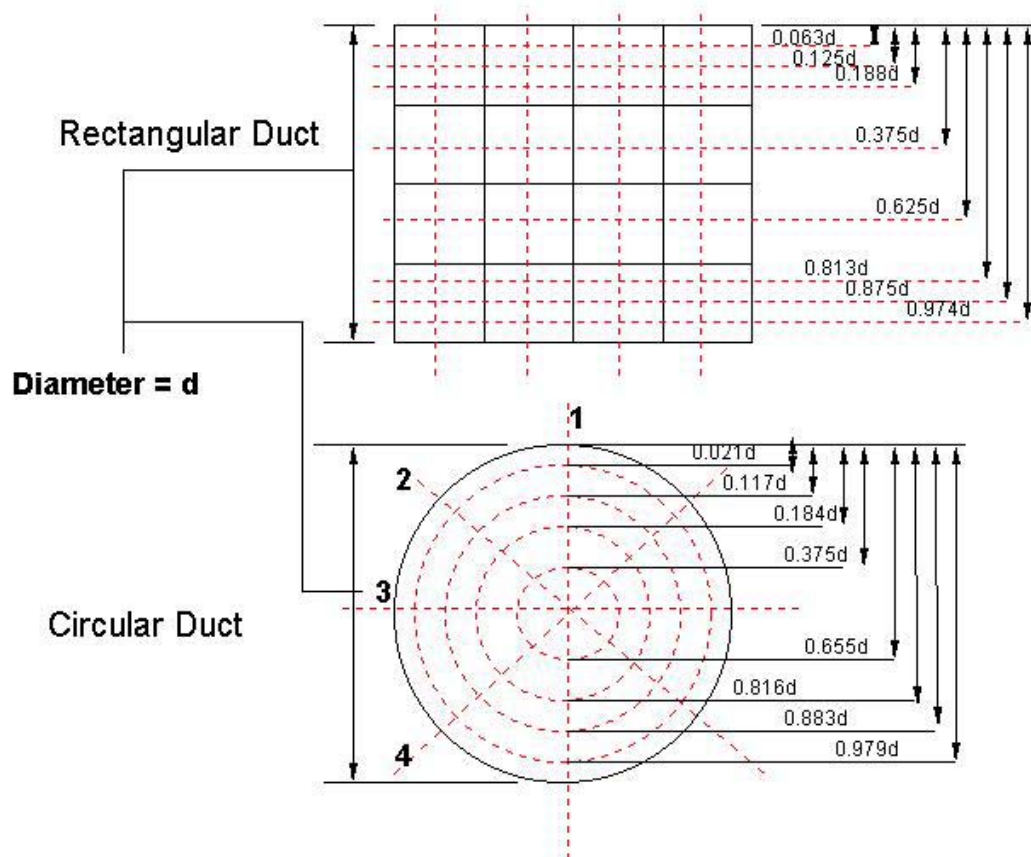
Fan ducts usually require the use of a pitot static tube, as velocities can be greater than 25m/s.

The velocity at a point is calculated from the equation for velocity pressure $v = \sqrt{\frac{2VP}{\rho}}$

The velocity in a duct will vary from point to point and the more points measured (on a designed basis) the more accurate is the result.

The duct is divided into equal areas and a measurement made at the centre point of each area. The Velocity at each point is calculated and the arithmetical average of all sections is determined and used as the duct velocity. The airflow quantity is determined by $Q = v * A$

The figure below shows measuring sites in circular and rectangular ducts derived from the British Fan Test Code.



The Australian Standard AS 2936 does not identify sites for rectangular ducts and for circular ducts shows 8 or 10 measuring sites on 3 diameters, as opposed to the 4 diameters in the British Standard. The 8-point measuring system is for general use with the 10 measuring point system used for swirling flow in circular ducts

Measurement Distances for Pitot Static Tube in Circular Ducts

8 Point System	10 Point System
0.021d	0.019d
0.117d	0.077d
0.184d	0.153d
0.345d	0.217d
0.655d	0.361d
0.816d	0.639d
0.883d	0.783d
0.979d	0.847d
	0.923d
	0.981d

Note d = diameter

6.16 Fan Laws

Fan performance curves are determined for a specific speed and density.

If density for a fan curve is not shown it can be assumed as standard air (1.2 kg/m³). Standard air is a definition of convenience that is widely adopted. The actual values of temperature, pressure and moisture content are not important.

Should the diameter, speed or density of the fan inlet air be altered, the following fan laws will apply.

6.16.1 Density (ρ) Change

Quantity (Q)	$Q_1 = Q_2$
Pressure (P)	$\frac{P_1}{P_2} = \frac{\rho_1}{\rho_2}$
Power (kW)	$\frac{kW_1}{kW_2} = \frac{\rho_1}{\rho_2}$
Efficiency (η)	$\eta_1 = \eta_2$

6.16.2 Speed Change

Quantity (Q)	$\frac{Q_1}{Q_2} = \frac{\text{rpm}_1}{\text{rpm}_2}$
Pressure (P)	$\frac{P_1}{P_2} = \left(\frac{\text{rpm}_1}{\text{rpm}_2} \right)^2$
Power (kW)	$\frac{kW_1}{kW_2} = \left(\frac{\text{rpm}_1}{\text{rpm}_2} \right)^3$
Fan total efficiency (η)	$\eta_1 = \eta_2$

6.16.3 Diameter Change

When the diameter (**D**) is increased and the fan is geometrically similar the following laws apply:

Quantity (Q)	$\frac{Q_1}{Q_2} = \left(\frac{D_1}{D_2} \right)^3$
Pressure (P)	$\frac{P_1}{P_2} = \left(\frac{D_1}{D_2} \right)^2$
Power (kW)	$\frac{kW_1}{kW_2} = \left(\frac{D_1}{D_2} \right)^5$

If the size difference is great then it can be expected that the fan total pressure may be larger than that given by these laws (scale effect).

It is also possible that a change in density due to compression of the air will occur affecting the results from the use of these laws. This is normally only of concern at pressures above 2.5 kPa.

This is discussed more fully in “Woods Practical Guide to Fan Engineering” or “Fan Engineering” published by the Buffalo Forge Company.

These laws should be used with caution as they are not applicable to all fans.

Example

A 1000m diameter fan tested at 1380 rpm and an inlet airflow density of 1.16 kg/m³ gave the following results,

$$\begin{aligned}\text{Quantity (Q)} &= 20 \text{ m}^3/\text{s} \\ \text{Pressure (P)} &= 1520 \text{ Pa} \\ \text{Power (kW)} &= 40 \text{ kW}\end{aligned}$$

What is the expected operating performance at 1470 rpm at standard air inlet density of 1.2 kg/m³.

From equations in sections 6.16.1 and 6.16.2

Note that density change has no effect on the quantity.

$$\frac{Q_1}{Q_2} = \frac{\text{rpm}_1}{\text{rpm}_2}$$

$$\begin{aligned}\text{Therefore } Q_2 &= 20 \times \frac{1470}{1380} \\ &= 21.3 \text{ m}^3/\text{s}\end{aligned}$$

The pressure change can be calculated by combining the effects of density and speed changes.

$$\frac{P_1}{P_2} = \left(\frac{\text{rpm}_1}{\text{rpm}_2} \right)^2 \quad \text{and} \quad \frac{P_1}{P_2} = \frac{\rho_1}{\rho_2}$$

$$\text{Therefore } \frac{P_1}{P_2} = \left(\frac{\text{rpm}_1}{\text{rpm}_2} \right)^2 \left(\frac{\rho_1}{\rho_2} \right)$$

$$\begin{aligned}\text{And } P_2 &= 1520 \times \left(\frac{1470}{1380} \right)^2 \left(\frac{1.2}{1.16} \right) \\ &= 1790 \text{ Pa}\end{aligned}$$

The change in power change can also be calculated by combining the effects of density and speed changes.

$$\frac{\text{kW}_1}{\text{kW}_2} = \frac{\rho_1}{\rho_2} \quad \text{and} \quad \frac{\text{kW}_1}{\text{kW}_2} = \left(\frac{\text{rpm}_1}{\text{rpm}_2} \right)^3$$

$$\text{Therefore } \frac{\text{kW}_1}{\text{kW}_2} = \left(\frac{\text{rpm}_1}{\text{rpm}_2} \right)^3 \left(\frac{\rho_1}{\rho_2} \right)$$

And

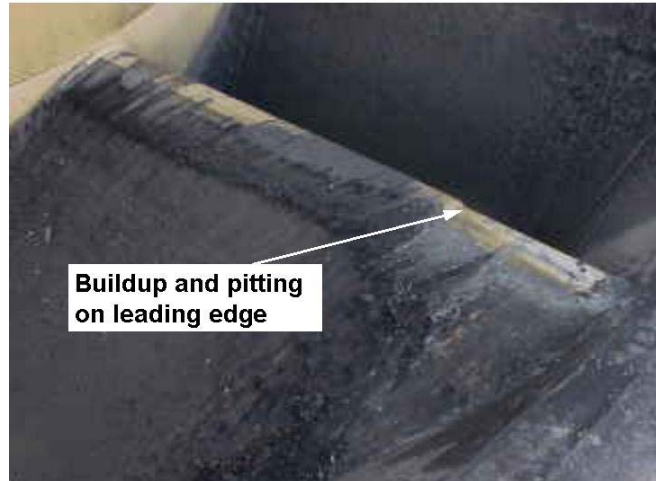
$$\text{kW}_2 = 40 \times \left(\frac{1470}{1380} \right)^3 \left(\frac{1.2}{1.16} \right)$$

$$= 50 \text{ kW}$$

6.17 Fans Failure

Fan failure is more often than not caused by:

- a) Poor preventative maintenance including regular cleaning of guide vanes and impellers,
- b) Unrecognised increasing fan pressure caused by increasing mine resistance,
- c) Rocks and other foreign materials are often found in fan housings and simple things like rags, plastic etc. can become caught on an impeller,
- d) An accumulation of dust and salts on impellers may also cause the fan to run out of balance,



All of which cause the fan to run out of balance. If a fan is allowed to run for extended periods while out of balance it can potentially cause bearing failure and fatigue cracking in either the impeller, the shaft or the fan casing.

6.18 Effect of Reversal of Rotation

1. An **axial flow** fan will deliver a reduced volume of air in the opposite direction at a greatly reduced pressure and efficiency, particularly if aerofoil blades are used
2. A **centrifugal** fan will deliver a reduced volume in the same direction at a greatly reduced pressure and efficiency

There is often suggestion that fans can be reversed in times of emergency to reverse the direction of flow in the mine. Although this is true the fan performance is significantly reduced. This could be 70%.

Before attempting this in an emergency the final affect must be completely understood and more importantly tried, tested and evaluated long before it becomes part of any emergency procedure.

“It remains for me to speak of the ailments and accidents of miners and the methods by which they can guard against these, for we should always devote more care to maintaining our health, that we may freely perform our bodily functions, than to making profits.”

Agricola - “De Re Metallica” (1556)

7 MANAGEMENT

There has been significant change in underground metalliferous mine design and operations over the past ten to fifteen years.

Hoisting shafts are now rare in all but very deep, high tonnage operations. In most mines, all ore and waste is brought to the surface in diesel trucks. More diesel equipment per tonne of ore is therefore required.

The use of ancillary diesel equipment (e.g. cable bolting, mechanical scaling, charge-up secondary breaking and shot-creting equipment) is increasing.

There is a greater prevalence of mining methods that require little permanent development apart from the decline. This has made the establishment of appropriately placed primary ventilation infrastructure difficult and increased the reliance on secondary ventilation techniques.

The regulatory framework has changed and is now much less prescriptive. Instead of designing to satisfy minimum statutory requirements, there is now much more reliance on the principle of an employer's duty of care.

There has been a dilution of technical expertise brought about by a number of factors, including the recent rapid growth in the industry. As a result, there is an increasing requirement for ventilation systems to be simple, robust and reliable so that minimal intervention or management is required.

Mine ventilation practice has had to evolve to keep pace with these changes. And many long established rules of thumb are now no longer appropriate.

Of all the ventilation problems encountered in mines, the vast majority are caused by higher than acceptable concentrations of smoke, dust and heat.

Smoke reported during the shift is generally from poorly tuned and/or maintained diesel engines. Dust is almost always from handling of dry broken rock, drilling and raiseboring operations. Increases in temperature are due to change in ambient surface conditions or ventilation flows being reduced (changes due to rock temperatures should be well recognised long before they cause problems). Identification of the source of the contaminant provides the opportunity to control the problem at this point. However, solutions of this type are more often than not met with resistance from production and maintenance personnel. Most cases are resolved by a compromise between removing the source and increasing the airflow.

In those situations where the source cannot be successfully controlled, for whatever reason, then rescheduling of activities may remove the symptoms and allow work to continue.

Unfortunately it is only the most severe cases of undesirable conditions that are adequately dealt with and the others are allowed to continue because “we will be out of there soon”. As a consequence operators are usually only looking for a “quick fix”.

Of all the problems identified the solutions could quite easily be found in the “Top Ten Solutions” below

1. Holes in auxiliary ventilation duct

2. Excessive tees NOT tied off
3. Fan re-circulation
4. Poor fan selection
5. Leaking ventilation walls
6. Poor primary airflow distribution (caused by incorrectly adjusted and/or lack of controls)
7. Lack of dust control measures when loading and transporting broken dirt
8. Short-circuiting of air through stope voids, ore passes and disused vertical openings.
9. Activities and diesel equipment has outgrown the original ventilation system design.
10. Poor records of airways (particularly vertical openings), dimensions, development method, location and purpose

All are problems that could (and should) have been avoided by better planning and management.

7.1 Primary Airflow Requirements

There are many “Rules of Thumb” (ROT) and a few empirical equations that have been developed to estimate the airflow requirements for mines. Some are realistic and based on recent mechanised mines but many are unrealistic and based on older non-mechanised mines. All are based on past experience. Their use is acceptable as long as they are taken exactly for what they are that is an idea of the magnitude of the result that will be achieved by calculation. Some time the results will be surprisingly close to a calculated result and many times it will be way off the mark.

Determination of the primary airflow requirement is the starting point for ventilation design. The design airflow needs to be known before the required number, and size of surface ventilation shafts can be estimated.

Design primary airflow requirements are almost always under-estimated, probably due to the use of ROT or some empirical equation, but also due to a failure to allow for increases in mine size, depth and complexity beyond initial estimates. Even then the cost of providing the necessary airflow under-goes “pruning” in an effort to keep project capital and operating costs to the minimum. Under-estimated (or reducing) primary airflow rates will adversely affect the project economics in the following ways:

Operating Costs: Power costs increase dramatically with airflow rate. If airflow rates need to be increased beyond initial estimates, there will be a substantial increase in operating costs, particularly if airways are inadequately sized:

$$\text{Power Cost Increase} \propto \left(\frac{\text{New Flow Rate}}{\text{Old Flow Rate}} \right)^3$$

Capital Costs: An increase in airflow rate may require the purchase of new fans and excavation of new airways. The cost and just as importantly the disruption to mining in doing so are usually considerably greater than would be the case if the original airways sizes were based on realistic primary airflow estimates.

7.1.1 Determining Primary Air Quantities.

When estimating primary airflow requirements, we need ask ourselves why ventilation is being provided in the first place. In many cases, in mechanised mines the criterion for dilution of diesel exhaust emissions is the overriding factor. It is however also necessary to consider equally

relevant requirements such as control of thermal conditions and adequate dilution of dust and mine gasses.

Analysis of airflow requirements for individual work places is usually carried out when determining primary airflow requirements. Design airflow rates are often based on exceeding 0.04 to 0.06 m³/s of airflow per rated kW of diesel engine power in all active mining areas. This analysis can become quite complex for larger operations and requires a thorough understanding of the individual facets of the development and production process. The design airflow rate must include an allowance for leakage (both primary leakage and leakage in secondary ventilation ducts) and service areas such as ore passes, grizzlies, crushers, conveyor drives, fuel bays, pump stations, workshops etc.

The first process is to work with the designers and schedulers to allocate air quantities and with consideration to predicted heat loads, strata gas emissions, diesel exhaust emissions, legislation, OH&S and Industrial relations to allocate airflow requirements for;

Location	Airflow required for
Scheduled production activity Drilling, blasting, loading, filling.	Equipment in use in each area Dimensions and layout of access and loading routes
Scheduled development activities	Equipment in use in each area Dimensions and layout of access and loading routes Auxiliary ventilation system
Haulage routes	Equipment in use in each area Dimensions and layout of access and loading routes
Rock handling including: Rock breaking Crushing, Loading station, Hoisting	Equipment in use in each area Dust dilution/collection/removal Operator protection
Servicing areas including: Maintenance workshops, Electrical sub- stations, Pumping stations, General maintenance and construction work areas, Refrigeration plants	Equipment in use in each area Dust & fume dilution/collection/removal Fire protection Operator protection
Storage and supply areas: Explosive, Diesel fuel, General consumables and stores	Equipment in use in each area Dust & fume dilution/collection/removal Fire protection Operator protection
Lunch rooms and waiting areas	Airflow Temperature

Obviously not all of the above locations/activities will apply to all mining operations.

Circuit design is ultimately determined by the production schedule.

The siting of dedicated ventilation airways will be determined by mining methods and production design layouts. The determination of total mine airflow is only the first step toward a mine ventilation system and requires the most attention because it is at this point that the final and cost to the operation will be determined.

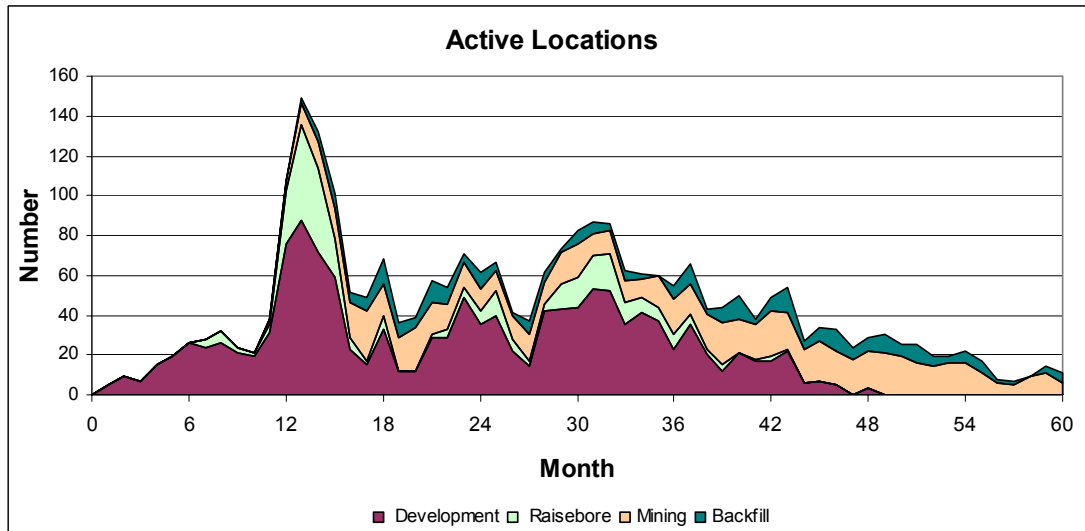
Ventilation planning and design is complicated and those who are responsible for ventilation and believe otherwise have the potential to cost their organisation a great deal of money.

Example

Lets consider a new narrow vein mine with an optimum production rate of 300,000 tonnes per year. It has been decided that predominately one pass ventilation will be required. Ore will be loaded from the stope to an orepass from where it will be trucked in the main decline out of the

mine. Backfill will delivered to the stope void using a 10 tonne truck. Assume a “ROT” for narrow vein mechanised mine to be $0.5 \text{ m}^3/\text{s}$ per 1000 tonne. ($0.5 \times 300 = 150 \text{ m}^3/\text{s}$)

The production and development schedule has been evaluated and the active locations during each month have been determined. In this case an active location is taken as any development face, raise bore location, production drilling location, production bogging location or stope filling location. The number of active locations during any one-month is then summed. For this example assume the results shown graphically below.

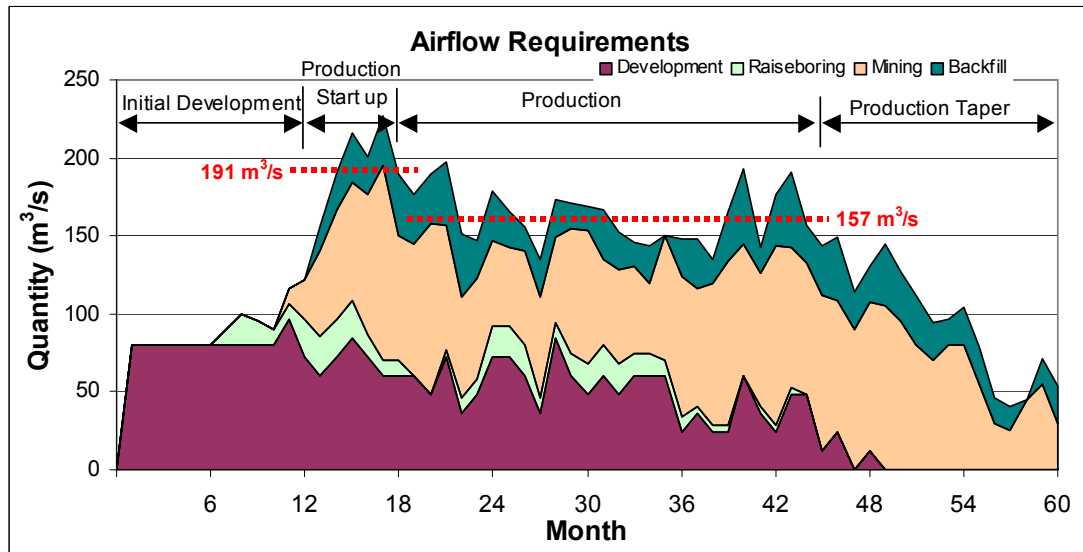


After consideration to diesel exhaust emissions, dust and heat etc. the air quantities for each activity have been determined as:

Activity	Airflow (m^3/s)
Decline development	80
In ore Development	12
Production	5
Raiseboring	5
Backfill	8

This airflow was then allocated to the respective activity and summed for each month.

This particular schedule will require primary airflow ranging from $80 \text{ m}^3/\text{s}$ during initial development peaking at $227 \text{ m}^3/\text{s}$ during the start up of production then tapering off to less than $50 \text{ m}^3/\text{s}$ toward the end of the scheduled period.



There are four distinct periods that require some comment.

1. Initial development (month 1 to 12) to establish the access decline and the exhaust ventilation shaft will be ventilated with secondary fans and will not form part of the primary ventilation requirements.
2. Production Start-up (month 13 to 18). The peak airflow during this period is 227 m³/s. This peak at the end of initial access development and the commencement of production is common. This occurs because of the flurry of activities necessary to bring the mine into full production in as short a time as possible.

The difficulty of this period is the cost for providing extra air when it is not required over the longer term. Most designers opt to bypass this period and attempt to manage their way through it. (Not a good time to be the ventilation officer). The alternative is to purchase and install primary fans with some form of control that will enable the airflow requirements to be “turned down” after production settles down.

3. Main production period (month 19 to 46). In a new mine there will be ‘teething problems’ and the production and development schedules will undergo constant updating before eventually settling down. With input from the ventilation engineer the airflow peaks and troughs will be smoothed settling somewhere around the median of 160 m³/s. (Always round UP its hard enough to get so don’t give it away).
4. End of the life of the project (month 47 onwards). A detailed mining schedule is only valid until the next update. Any schedule beyond three years should be viewed with some scepticism. Because of increasing geological information, changing economics etc most mine schedules beyond year three do not have the same confidence level contained in the three-year plan. Because of the very nature of mining there will be an end and the tapering off period will vary from mine to mine, and year to year.

7.2 Primary Ventilation Fans

Primary ventilation fans are found in many different sizes, types and locations. For example they could be any combination of

- Forcing or exhausting.
- Axial or centrifugal.
- Single or multiple fans.

- Single or multiple shafts.
- Horizontal or vertical mounted.
- Horizontal or vertical discharge.
- Surface or underground.

Each has their advantages and disadvantages.

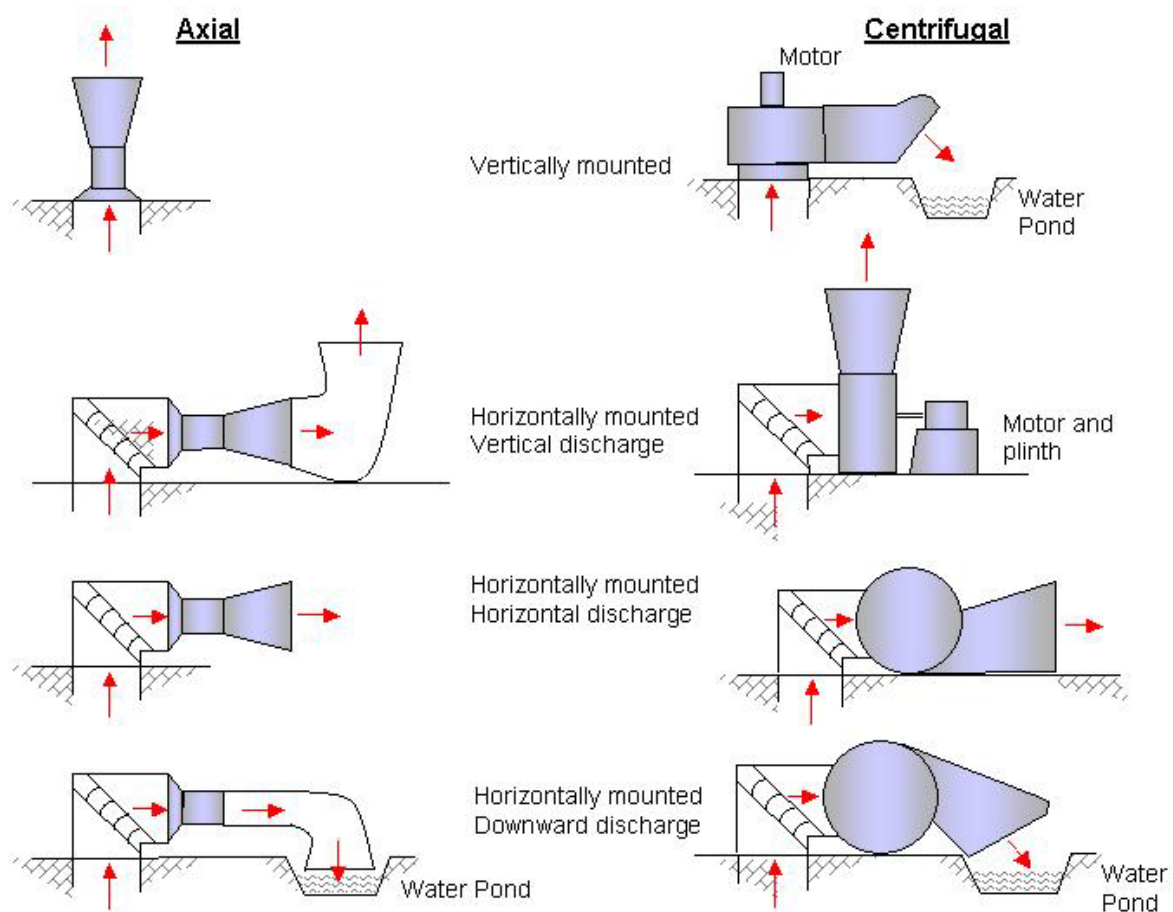
Some are designed to meet predicted requirements and some are found lying around and installed in the hope that they will provide the necessary airflow.

The preference of any particular combination is usually a matter of individual choice driven by past experience, environmental issues and capital cost.

Most fans are selected to meet the immediate short-term budget and not the long-term future ventilation requirements of the mine. Because of this they are not suitable for the life of the mine and have to be either upgraded or replaced both very expensive options.

7.2.1 Surface Fan Installation Arrangements

Primary ventilation fans are arranged in many different ways some of which are depicted in the figure below.



7.2.2 Underground Primary Fan Arrangements



Primary fans installed underground.

Primary ventilation fans may also be installed underground. The underground locations could be at the bottom of the shafts or at some intermediate point between the surface and the bottom of the mine.

Although underground fans installations are predominantly the axial flow type, centrifugal are also used.

The use of primary fans installed underground has generally been avoided for a couple of reasons:

1. Excavation and preparation of the site is costly.
2. If the fan fails access for repairs is restricted because the only ventilation flow is from the natural ventilation pressure.
3. Construction is more difficult as the use of large mobile cranes is restricted.

7.2.3 Forcing or Exhausting



Exhausting air from a shaft

men and equipment as it is much more desirable to enter the mine in clean intake air.

If the fan is installed in the intake airway it limits the accessibility to the shaft for the movement of men and equipment.

7.3 Circuit Booster Fans

Circuit booster fans are used to alter the pressure distribution and supply ventilating air to specific areas of the mine.

They are often necessary when a mine has

The vast majority of primary fans installed in Australian mines are located on the surface to exhaust air from the mine.

The choice of location is usually made as a matter of convenience rather than any specific engineering aspect. For example a simple mine will have one primary intake airway (be it shaft or decline) and one primary exhaust shaft.

Exhaust shafts are by their very nature dirty and the air contaminated with dust and fumes making it undesirable for the movement of



Forcing air down a shaft

been developed below the mining area defined in the original mine designs.

They are also misused in many mines as a method to supply air to areas of mines in an attempt to counteract the poor management of the primary ventilation circuit.

Poor selection of fan location and capacity has potential to create more problems that they solve. For example they may result in recirculation of large quantities of air, as they tend to pressurise the exhaust system forcing the air to flow back into the primary intake airflow.

7.4 Primary Ventilation Circuits

Primary ventilation circuits are designed to supply fresh air to workplaces and dilute and remove contaminants resulted from the mining processes. The time required to flush blasting contaminants from the mine can range from a few minutes up to many hour.

The effective and timely removal of contaminants particularly after blasting is high on the priority lists for mine managers, as time lost on production waiting for fumes to clear is very costly.

There are two basic circuits used

1. Parallel circuits, and
2. Series circuits.

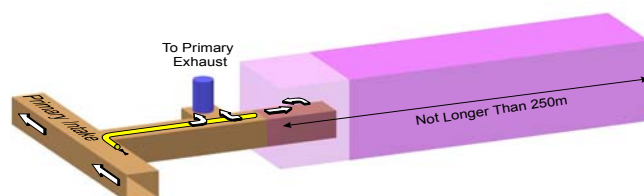
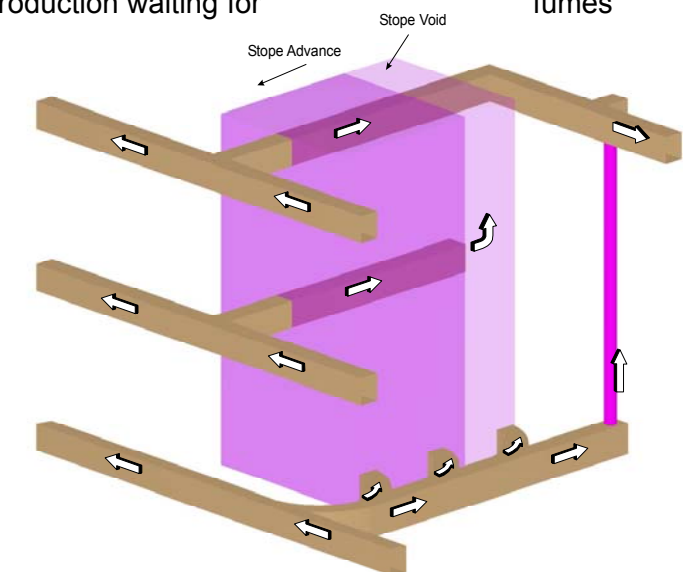
Both have their advantages and disadvantages and many mines use a combination of both circuit types.

7.4.1 Parallel (One Pass) Circuits

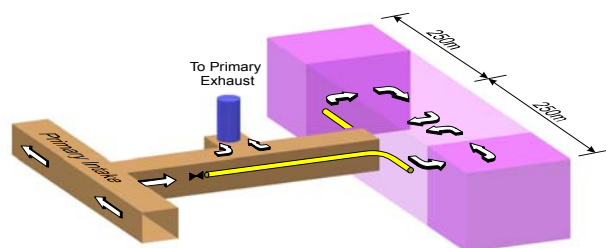
Parallel circuits adopt a general design philosophy of 'one pass, flow through' air.

This concept ideally lends itself to large tonnage stopes or stoping blocks and may not be suitable for use in some narrow vein steeply dipping orebodies. In this situation, a circuit is established to direct air from the primary intake circuit over the areas of activity and exhaust directly to the return air circuit. It is usual for return air circuits to be areas requiring minimal access and therefore all contaminants are removed directly from the mine. This is particularly important in the event of a fire occurring in the work place as it minimises the areas (and personnel) affected by the smoke, gases and particulates.

With multiple level accesses, it is necessary to control the air entering the activity area (usually by regulation of the exhaust). The down side of this



TYPE A: End Access



TYPE B: Central Access

type of circuit is that short-circuiting occurs each time a stope is opened or a ventilation control is damaged.

Whilst providing improved conditions in work areas, these circuits also decrease re-entry times but only if the access does not form part of the return air circuit.

It is usual to provide airflow to all scheduled production locations, even when not in use. This provides flexibility to move from one location to another without the necessity to adjust ventilation circuits. Mine operators often plan to minimise total airflow by campaigning air from one location to another, however experience tells us that this seldom, if ever, occurs and all locations operate with less than desirable airflow rates.

For this type of circuit to be successful, ventilation controls must be well maintained.

7.4.2 Secondary Fans and Parallel Ventilation Circuits.

Because of the geometry of narrow vein orebodies, it may prove difficult to maintain the philosophy of flow through ventilation without the need for extensive development and therefore the use of secondary fans during production must be considered.

This type of circuit is normally adapted to narrow vein orebodies with long strike lengths. LHD tramming distances are usually kept below 250m. In these circuits, air intakes via the access (decline) and is returned to the primary exhaust at each orebody access.

7.4.3 Series Ventilation Circuits

A system most often used in narrow plunging orebodies, is a simple series ventilation circuit. These circuits rely on secondary fans to ventilate in orebody development and production with the return air from these activities being mixed with the primary intake airflow and re-used at the next activity.

This type of circuit has the advantage of simplicity of control and minimal development for ventilation.

Although the utilisation rate (total airflow to the working areas / total airflow through the mine) can be as high as 75%, it is typically 65% with 40% not uncommon.

These circuits have disadvantages including (but not limited to):

- The need for high pressure fans (hence higher power costs),
- Low fan power efficiency for the majority of the operating period (the fan only operates at maximum efficiency in a fully developed mine).
- Decreasing airflow with depth, and
- Contaminants from activities (and fires) affect all down stream personnel.

Although not encouraged, it is recognised that this is the system used in development areas and in reality will continue to be used in some production areas.

7.4.4 Use of Stope Voids as Airways

There is often a misconception that stope voids will upcast airflow causing much frustration when the air downcasts through the stope.

Because stope voids form connections between levels they create airways parallel to the ramp or intake shaft. Generally the pressure distribution is such that the air in the ramp (intake system) is down casting and it follows that the air in the stope will also downcast air. Obviously if the air in the ramp is up casting then it follows that the air in the stope void will also upcast.

If stope voids are required to up cast air, then it is necessary to design and install the appropriate ventilation controls to cause the pressure distribution required. In general, if stope voids are required to upcast, the primary return airway must not be lower than the top of the stope void.

The reliance on stope voids also has the high probability of production blasting covering the stope brow thus preventing flow through ventilation.

7.4.5 Recirculation

Recirculation occurs when air is kept within a closed circuit. It should not be confused with the situation when air is reused, as in series ventilation circuits.

Recirculation occurs when a fan is installed in an airway in which the natural flow of air along the airway is less than the operating capacity of the fan. In some Australian legislation there is a requirement to install secondary fans such that the air delivered to the fan is greater than 1.3 times its open circuit capacity. This legislation was introduced in times when small drives (2.0m x 2.0m) and low capacity fans (5 to 7 m³/s) were common. As a general Rule of thumb an airflow of at least 1.5 times the open circuit capacity of the fan is required in airways with large cross sections and higher capacity fans. Even then minor recirculation is possible, depending upon the siting of the fan in the airway.

Although not recommended, some minor recirculation may be acceptable provided the, work place temperature, contaminants in the airflow and the clearing time for blasting gasses remain within acceptable levels.

7.5 Ventilation Controls

Regulators commonly used in Australian mine are rectangular openings in walls that are open or closed by the placing or removal of boards. Other less common regulators are the louvre type (either vertical or horizontal), or a sliding door (again either vertical or horizontal).

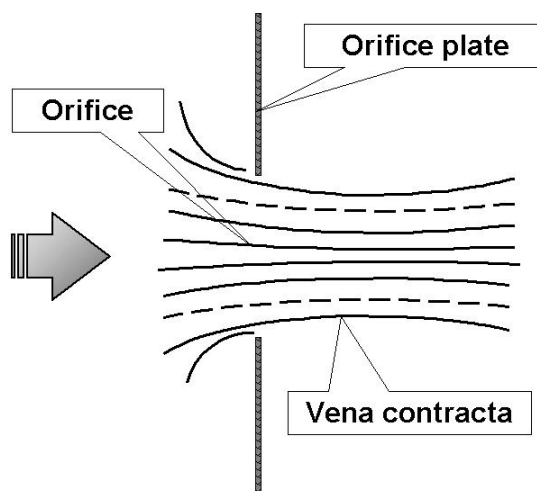
It is normal for persons to open or close a regulator in order to increase or decrease airflow to a certain area. Sometimes this is met with much frustration because the desired airflow cannot be achieved. This frustration is a result of the lack of understanding of the pressure distribution across the mine and the fact that altering one regulator effects the WHOLE mine distribution and not just the area being adjusted.

There is often an attempt to compare the opening in a wall to an orifice (vena contracta). In a perfect orifice the opening is circular and relatively small compared to the plate (wall), which is very thin. As the air flows through the opening it converges and at a distance equal to half the orifice diameter the jet is at its smallest area. This area is called the **vena contracta** and at this point the velocity of the air is 1.613 times greater than in the orifice itself. At the vena contracta the velocity pressure is 1.613² or 2.5 times higher than the orifice.

Using the equation for velocity pressure

$VP = \rho \frac{v^2}{2}$ and knowing the velocity we can determine the velocity pressure. Similarly if the pressure across the plate is known, then the velocity in the area of vena contracta can be determined and we can back calculate the area of the orifice.

Considering a mine regulator, the wall thickness can be proportionally greater



than the thin orifice plate and the ratio of the area of the opening to the area of the wall is different and this affects the coefficient of contraction (1.613) that becomes larger as opening increases in size forming an increase in percentage of the opening to the airway wall. For any calculation to be reasonably accurate the area of the regulator should be no greater than one-tenth the size of the airway in which it is constructed.

To estimate the area, quantity or pressure at a regulator the following algorithm can be used

$$A = 1.2Q\sqrt{\frac{\rho}{P}} \quad \text{Equation 5 Area of a Regulator opening}$$

Where A = the area of the opening (m^2)
 Q = Quantity of airflow (m^3/s)
 P = Pressure across the opening (Pa)
 ρ = Density of the air flowing (kg/m^3)

Like the calculation of shock losses in practice there are so many unknown factors that the calculated result is seldom correct and further adjustment is required. In practice the airflow through regulators is set by trial and error until the desired, or at least close to the desired result is achieved.

It is often the case that operators, to gain increased airflow, adjust regulators. Although they achieve their desired result it is more often than not that they have caused a detrimental effect else where in the mine.

All adjustment to regulators should only be undertaken once the airflow distribution across the mine has been planned and scheduled as it may be necessary to adjust more than one opening to achieve the desired result for all areas. The suggested method for any adjustment programs it to go to the bottom of the mine (lowest pressure area) and fully open this regulator. All other adjustments are then made to this opening. Commence adjustment (setting the desired airflow) at the top of the mine (highest pressure) and progress to the bottom of the mine (lowest pressure). By the time you get to the lowest regulator a measurement of the airflow should show that the airflow is around the quantity required.

7.6 Multiple Access Orepasses

Multiple access and uncontrolled orepasses will, when empty, form an airway parallel to the decline potentially reducing the airflow on the decline to unacceptable levels. In addition to short-circuiting airflows, multiple access orepasses are often a significant source of dust. As rock falls down the pass it generates dust and acts like a pump compressing air in the pass and forcing it out on to each open sub-level below. Efforts to prevent this by creating a positive airflow into the ore pass from all access points are generally unsuccessful. The problem is best managed by sealing, if necessary with temporary seals, all but one of the tipping points. ALL other openings should be sealed or otherwise connected to the RAR system via a dedicated airway.

7.7 Secondary Ventilation Systems

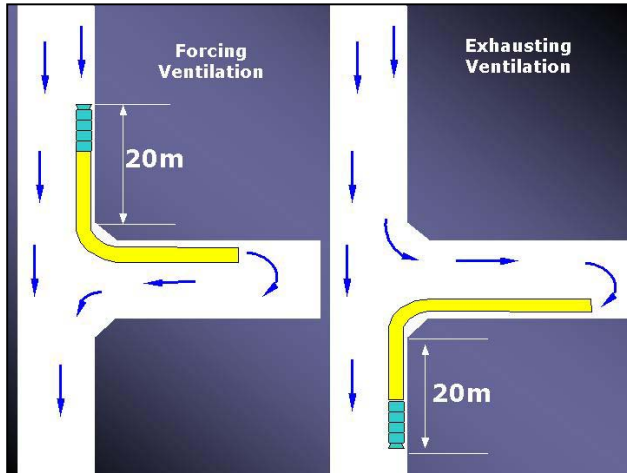
There are three predominant secondary ventilation systems used.

1. Forcing,
2. Exhausting and
3. Exhaust overlap

Forcing ventilation system is the predominant system used in hard rock mines. These systems provide advantages including

- The fan is located in clean intake air making it accessible for maintenance for maintenance and repair
- Allows the use of flexible ventilation ducting (less expensive than rigid ducting)

Exhaust ventilation systems require the use of rigid ducting and because of the extra cost involved with the purchase and installation of rigid ducting, systems of these types are seldom used in development headings in hard rock mining. However, it is the accepted practice in coalmines and many tunnelling operations that use cutting or boring machines.

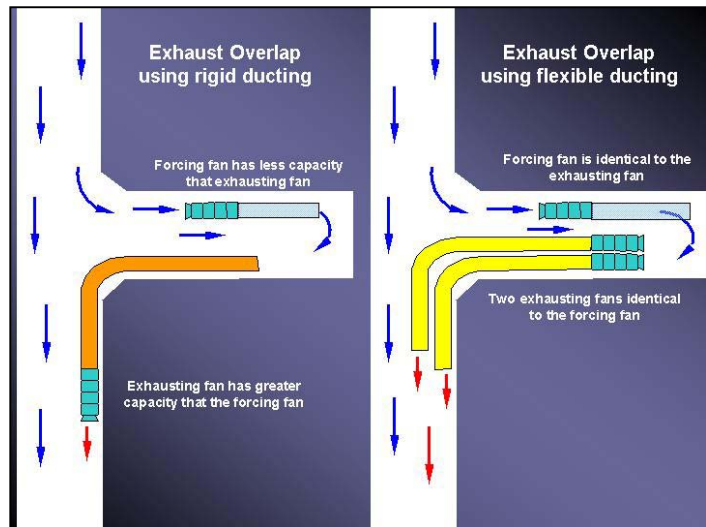


Re-entry times may be reduced by the use of an exhausting or **exhaust overlap** ventilation system where the air is removed via the ducting allowing immediate re-entry to the face in fresh air. Although not commonly used this system has benefits when developing long drives.

In its original concept the exhausting fan was slightly larger than the forcing fan, was installed clean air and used rigid ducting. This allowed air to flow in the overlap position and prevented re-circulation. Because of the cost of rigid ducting as opposed the less expensive flexible type this arrangement lost favour.

In recent times it has been adapted for use using flexible ducting and two exhausting fans. The use of two exhausting fans was simply to standardise the fans being used. Because the fans are located in the developing heading if there was to be a

power failure to both fans then repairs are restricted until the area is ventilated by another fan. It is also inconvenient to move the position of the overlap as it is necessary to remove and relocate the exhausting fans.



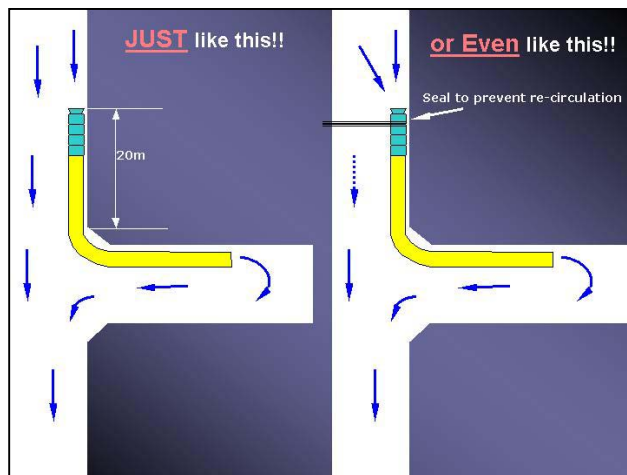
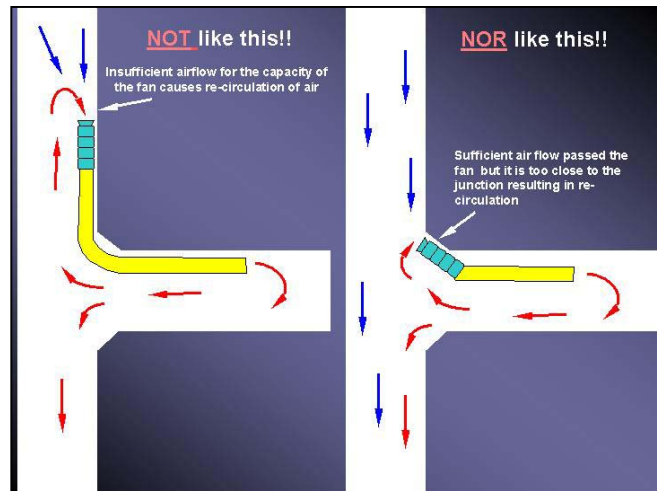
7.7.1 Installing Secondary Ventilation Fans

Auxiliary ventilation fans are a fact of life and used in ALL mines yet they are still installed where it is convenient rather than where it is best suited. Poor location of auxiliary fans will result in recirculation of air and obviously recirculation of contaminants with the resultant "less than desirable" conditions at the working face. In most cases once the fan is installed it is seldom relocated, no matter how "bad" conditions become. There have been instances where fans have been poorly located to the extent that the recirculation is almost 100%. This can go undetected simply because the fan "is running" and you can feel the breeze at the face.

The ultimate reason that fans recirculate is the lack of air flowing passed the fan inlet. This can occur for three reasons:

1. There is insufficient airflow in the drive,
2. The fan is too close to the exhaust
3. The seal around the fan is damaged.

One of the simplest means for detecting recirculation is to use a smoke tube (particularly near the backs) to check the direction of the exhausting airflow. Sometimes it is extremely difficult to get close enough to the backs and it will be necessary to undertake measurements of air flowing in the drive and in the duct downstream of the fan. In all cases the airflow in the drive must be greater than the airflow in the duct.



In general auxiliary fans should be located approximately 20m upstream of the drive they are ventilating, and there is a “rule of thumb” that the airflow passed the fan should be 1.3 to 1.5 times the open circuit capacity of the fan. If one or both of these factors does not apply the fan must be sealed into and appropriate airway to prevent any re-circulation.

Many mine designs now incorporate the return air rise (RAR) adjacent to the intake air decline. This allows contaminated air to be exhausted from the mine without being reused. (See “Secondary Fans and Parallel

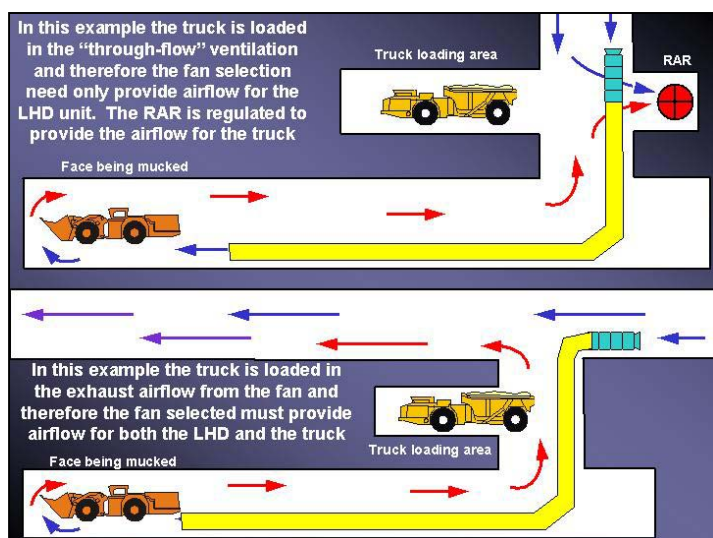
Ventilation Circuits” above). In many of these cases the truck is loaded near the stockpile bay adjacent to the RAR and care must be taken to ensure that there is a flow of air in this area. In the same way that the exhausting fans are sized to cause a flow in the overlap, a flow must be caused between the fan and the exhausting point. If this is not achieved the truck loading area very quickly becomes contaminated with high concentrations of dust and diesel exhaust emission.

Some options are shown in the figure opposite.

7.7.2 The “Reuse” of Air

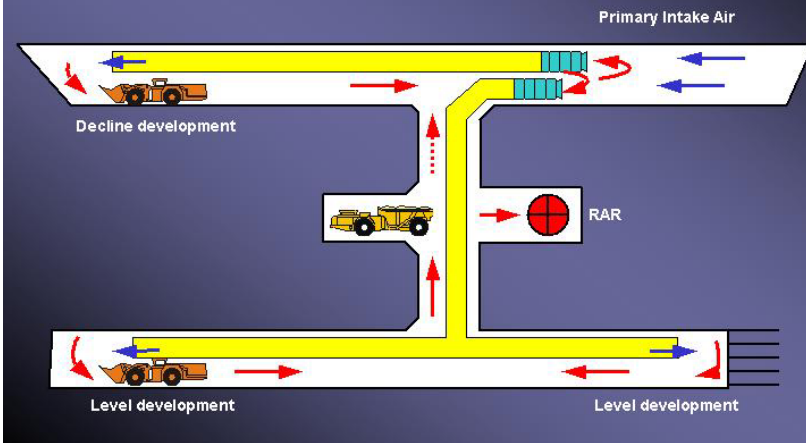
As discussed above, it is an acceptable practice to “reuse” ventilating air. The success or otherwise of this practice in force ventilated areas lies in the correct selection and location of secondary fans.

Many attempts are made in development areas to install multiple



In this example two fans have been installed to ventilate two areas, but the primary intake air is less than the capacity of the fans therefore the fans “re-circulate” some of their air resulting in increased atmospheric contaminants.

The obvious solution is to increase the primary intake airflow but sometimes this is not possible and one of the fans MUST be relocated to reuse the air.



parallel fans in an effort to get extra fresh air to the working faces. In the vast majority of these installations the airflow passed the fans is insufficient, resulting in the “recirculation” of the return air through the fans. The final result is increased heat and contaminants in the working areas.

There are a number of solutions the obvious ones being, increase the primary intake air, only operate one fan at a time, construct a seal around the fans to prevent the recirculation. Only increasing

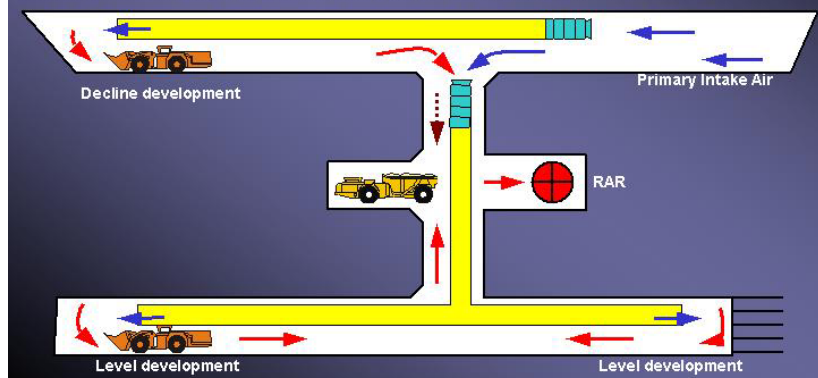
the primary airflow would not restrict the flexibility of multiple workplaces or cause some interference with activities in the area.

Another solution is to relocate one of the secondary fans.

In this example two fans have been installed to ventilate two areas but the primary intake air is less than the capacity of two fans but greater than the capacity of one fan.

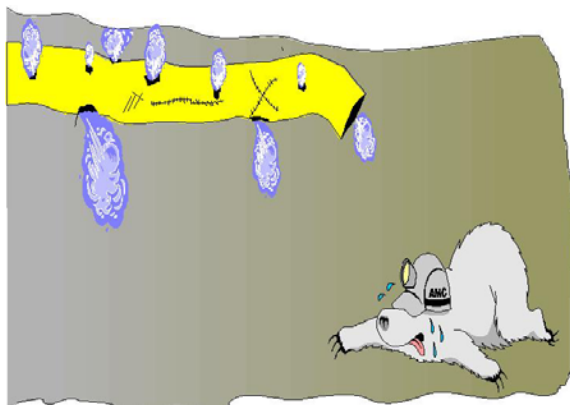
Installing the fan between the access and the RAR will allow the primary intake air to be mixed with the return air from Decline development before or as it enters the fan for the level development.

When ‘reusing’ air in this manner monitoring of contaminants is necessary to ensure contaminants do not increase above acceptable concentrations.



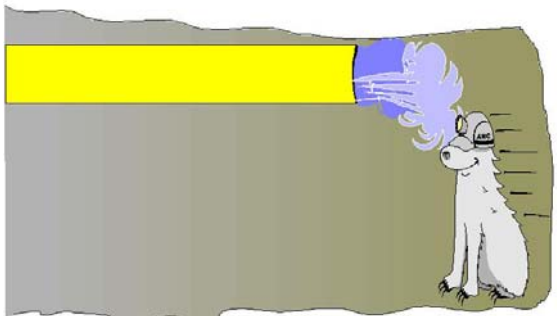
7.8 Ventilation Duct.

Flexible ventilation duct is an extremely versatile and convenient method of providing secondary ventilation. Unfortunately, its limitations are very poorly understood. Some practical tips for installation include:



- The largest possible ventilation duct size should be used to lower resistance and hence reduce leakage. In almost all cases, using **duct size which is smaller than the fan diameter is not acceptable.**
- No one seems to get particularly concerned with **holes and rips** in ventilation duct. Like

compressed air leaks, the very high cost of these problems are often not appreciated, or are simply ignored. It is a common experience to almost have one's head blown off by leakage from a ventilation duct near the fan and to then be asked why there doesn't seem to be much flow at the face.

- The importance of ensuring **adequate clearance between mobile equipment and the duct** is also not clearly understood. This wasn't the case in the old days when steel duct was used! The equipment and drive size *must* be selected so that there is *no* chance of the duct being hit.
 - The importance of **installation standards** is also not often appreciated. The knocker line must be hung straight from lined up attachment points (not from the nearest convenient split set). Sharp corners require the use of specially manufactured elbow pieces. Every eyelet must be used when connecting "Lo k®" or similar joints. Care must be taken to ensure that "Minsup®" clips are not left in the ends of ventilation duct when new bag is connected. The last one or two ventilation ducts should be sacrificial "face bags" to avoid the rest of the duct sustaining blast damage.
- 
- There seems to be a common belief that an **unlimited number of "T" pieces** can be connected to a ventilation duct and left open. This is incorrect. There is usually only sufficient flow to ventilate *one* face, possibly two and all other "T" pieces should be tied off.
 - There is a perception that "more air" can be supplied to the face by using a more powerful fan. With electrical power costs of 10¢ per kWh, and 170 kW power draw, a fan would consume \$149,000 per annum in power costs. It is not unusual to find 1400 mm diameter fan and duct installations, which over 500 m deliver only 7 m³/s to the face! This is insufficient flow for even one LHD. For comparison, 23 m³/s can be delivered, with very well installed 1,200 mm low leakage ducting using a 1,200 mm diameter, 110 kW fan.
 - Use of the low leakage ducting described above can also enable large amounts of capital to be saved. With the use of low leakage, well installed flexible ventilation ducting it is possible to advance development headings in excess of 2000 m. This may result in the savings of many millions of dollars by not having to excavate ventilation airways at various points along the development heading.

7.9 Duct Leakage

Generally, the prime consideration for the purchase of ventilation ducting is cost, not only with the original purchase price but also with the cost of transport and handling. Secondary to this is the ease and convenience of installation and thirdly there may be occasional consideration of the material friction factor but very seldom is any consideration given to the rate of leakage from the duct.

Prior to the 1960's the ventilation ducting of choice was usually a rigid type manufactured from galvanised iron or plastics of some type.

Lay-flat flexible ventilation ducting has been used extensively in Australian Mines since around the middle of the 1960's. In it's original form it was manufactured from a terylene material with wire hoops sewn into each end. These hoops were overlapped one inside the other and prevented from coming apart by tying a length of wire around the duct in between the hoops.

Prior to the introduction of trackless diesel equipment auxiliary fans had relatively low airflow rates (6 to 7 m³/s) and low pressures (2000Pa). Since the late 1960's and the introduction of diesel powered trackless equipment, airflow requirements for auxiliary fans have increased and as a consequence so have fan pressures. In response to a need to lower operating costs in the 1970's different materials were introduced as was a different type of joint (spigot type). This development process has introduced three potential areas of leakage, the material, holes created for stitching the seams and, the type of joint.

The requirement for increased airflow has grown to the extent that auxiliary ventilation fans now provide up to 50m³/s at pressures in excess of 4000Pa resulting in increased leakage quantities. The air that leaks from a duct is a function of many variables including; type of material, type of joint, performance of the fan, and the number and size of holes in the duct. The combined affect is reduced airflow discharged to the working face.

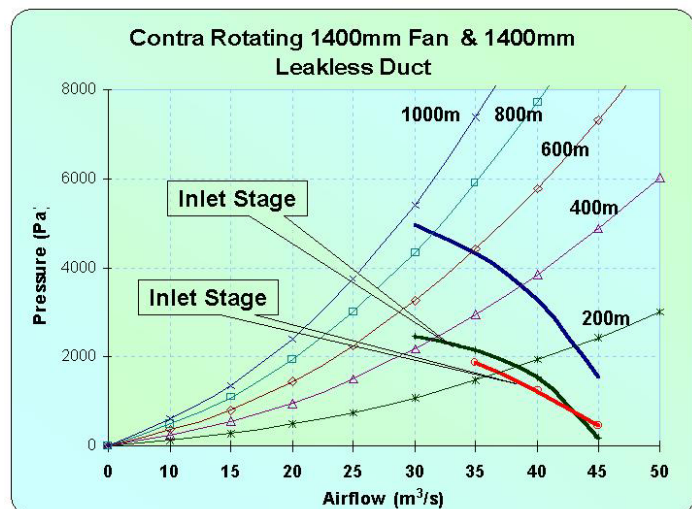
Many of the early development problems have been recognised and material coatings have been improved. Stitching has been replaced with welding and the hoop type joint has been resurrected and improved by holding it together with a specifically designed clamp. These improvements have all come at increased purchase price.

The high leakage rates from some spigot low cost ducting currently available in Australia limits its use to lengths of up to 400m in some cases this can be as little as 200 m. After this point leakage from the system can be as high as 50% of the air produced by the fan. The leakage rate in a well installed and well maintained low cost ducting may cause only slight inconvenience in short (less than 400m) development headings and the use of the lower cost stitched, ducting and the spigot joint continues to serve this purpose. Because most mines have spare development fans any excessive leakage is "fixed" by installing an additional fan either in series or parallel.

Although ventilation ducting with a high leakage factor may be considered suitable for short development headings, its use in long development headings is questionable. There are some who believe that the introduction of extra fan power (i.e. another fan in the duct) will deliver the required quantity of airflow to the face.

In Section 6.3 page 93 we showed how we could predict the airflow through a system using a fan performance curve and the system resistance curve. Calculating the resistance curves for various duct lengths can also allow us to predict the airflow achievable at various lengths. The shortfall in this methodology is that it assumes "leakless" ducting.

In the example shown opposite a two-stage contra rotating fan in a "leakless" duct will provide 43 m³/s at 200m and 32 m³/s at 800m. In reality ducts are seldom (if ever) free of leaks and this is particularly true of flexible, lay flat types of ducting, and as a consequence the pressure requirement is reduced and the stall point is seldom reached. With leakage in a good quality ducting the airflow through the fan at a distance of 800m, would be slightly higher (say 38m³/s) and the flow to the face would be significantly less and would probably be closer to 20m³/s, and is barely adequate for the operation of one truck and one LHD.



The air that leaks from a duct is a function of many variables including; type of material, type of joint, performance of the fan, and the number and size of holes in the duct. The combined affect is reduced airflow discharged to the working face.

The other important point to note is the air through the fan has not significantly changed and is operating very low on its performance characteristic curve. With this in mind the location of the fan becomes important if recirculation is to be avoided. As discussed earlier a general 'rule of thumb' for auxiliary ventilation fans is to locate them in airflow that is equal to 1.5 times the open circuit capacity of the fan. In the case of a 1400mm diameter fan this is $67.5 \text{ m}^3/\text{s}$ (1.5×45). If this airflow is not available then it may be necessary to install a seal around the fan to prevent any recirculation.

Vutukuri (1984)¹⁸ developed a methodology for use in a computer program necessary to solve the complexity of two components:

- Duct friction resistance
- Duct leakage resistance

Without leakage, determining duct friction resistance is straightforward enough. It can be determined simply using Atkinson's equation. Similarly, consideration of duct leakage resistance on its own is relatively simple. The leakage resistance is often expressed as a leakage flow per unit length for a given duct diameter at a given pressure, or more intuitively, as an equivalent mm^2 of holes per m^2 of duct.

The problems commence when the interaction of the above two resistances is considered. In particular, determining fan pressure requirements for a given flow is difficult, because the flow rate along the duct is not constant.

The only constant in ventilation duct system is the fact that it will leak but how much is anyone's guess.

7.10 Velocities in Primary Airways

The economic optimum diameter of any ventilation airway can be determined by the total development and operating costs over the useful life of the airway. Other engineering factors include;

- dust entrainment,
- cooling power of the air,
- pressure losses caused by conveyances (cages in shafts, conveyors in drives, trucks in declines etc.),
- effect on equipment in the airway (harmonics in rope guides in hoisting shafts),
- activities being undertaken in the airway,
- type of equipment to be used for development of the airway, or
- any combination of these and other factors.

7.10.1 Velocities in Access Drives

There is a generally accepted "guideline" for a maximum air velocity of 5.0 m/s to be used in declines and other access drives. The purpose of this is to ensure that dust is not entrained in

¹⁸ VUTUKURI, V.S. "Design of Auxiliary Ventilation Systems for Long Drives" Fifth Australian Tunnelling Conference, Sydney (1984)

the primary airflow and that unnecessarily high ventilation pressures (and consequently high electric fan power costs) are not imposed on the mine ventilation system.

Velocities above 6.0 m/s can be cost effective over short distances providing that good dust management systems in place to ensure that:

- dirt in the trucks is sufficiently wetted, and
- roadway surfaces are sufficiently stable (wetted) to prevent dust pick up from tyres being entrained into the ventilating air.

7.11 Water in Upcast Shafts

The processes of mining rely on the use of water for dust suppression and air-cooling. As the ventilating air flows through the mine workings it takes up this water as vapour until it becomes, to some degree, saturated. As this saturated air rises from the bottom to the top (surface) of the exhaust shaft the barometric pressure is constantly decreasing and as a consequence the temperature of the air, and water vapour mix, is also decreasing. This results with water condensing out of the air.

This condensation process is evident by a 'foggy' appearance of the air. This 'fog' is small droplets of water that may coagulate to form larger droplets.

A large water droplet of greater than 3mm diameter has a terminal velocity of 8.0m/s. If the velocity of the air is less than 8.0m/s then the water droplet will fall down the shaft and if the velocity is greater than 8.0m/s the water droplet will be pulled up the shaft. Obviously if the velocity of the air is 8.0 m/s it stands to reason that the water droplet will neither rise nor fall and will remain suspended at this critical velocity.

As droplets accumulate they form a 'water blanket' that increases the resistance in the shaft thus lowering the quantity of air flowing. At times this may be sufficient to cause the fan to go into stall. As the stall point is reached the water blanket drops out and the fan may recover to recommence the cycle of water blanket formation and drop out. If allowed to continue unchecked this may eventually cause mechanical failure of the fan.

Although 8.0m/s is seen as the critical velocity this is not an exact figure and will vary according to a number of factors. Water droplets may coagulate to larger droplets, and the air velocity in the shaft will vary inversely according to the density. This results in a 'critical range' of air velocity between 7.0m/s and 12.0m/s in which this phenomenon occurs.

When designing exhaust ventilation shafts in deep mines with potential for saturated air the velocity range between 7.0m/s and 12.0m/s should be avoided.

The effect is not as pronounced in shallow dry mines but the 'critical velocity range' should be avoided if there is potential for ground water to be present in the shaft.

7.12 Equipment Movement in Underground Airways

Trucks (or other vehicles) travelling in underground drives can cause pressure losses up to 200Pa depending upon a number of variables such as the size of the vehicle in relation to the cross sectional area of the drive and the direction of the airflow in the ramp. Consider the following:

A truck having dimensions of 3.1m high x 3.5m wide is travelling at 20kmph against an airflow of 100 m³/s in a drive 5.5 m high and 5.2 m wide.

The shock loss pressure drop can be calculated from

$$P_{\text{Shock}} = \frac{\rho}{2} \times \left(\frac{y(V_A - V_T)}{1 - y} \right)^2$$

Where: P_{Shock} = the shock loss pressure drop (Pa)
 ρ = the density of the air (kg/m^3)
 $y = \frac{\text{Area of the truck (m}^2\text{)}}{\text{Area of the Drive (m}^2\text{)}}$
 V_A = Velocity of the air in the unobstructed drive (m/s)
 V_T = Velocity of the truck relative to the airflow (m/s)

Assuming standard air density of 1.2 kg/m^3 then the pressure drop caused by the truck is 185 Pa.

i.e. Area of the truck = 18.85 m^2
 Area of the drive = 28.6 m^2
 $y = 0.659$
 $V_A = 3.5 \text{ m/s}$
 $V_T = -5.56$ (- since the truck is travelling against the airflow) m/s

Substituting these values into the equation:

$$P_{\text{Shock}} = \frac{1.2}{2} \times \left(\frac{0.659 \times (3.5 - -5.6)}{1 - 0.659} \right)^2$$

$$= 185 \text{ Pa}$$

In the same situation and only $25 \text{ m}^3/\text{s}$ the pressure drop would be 94 Pa

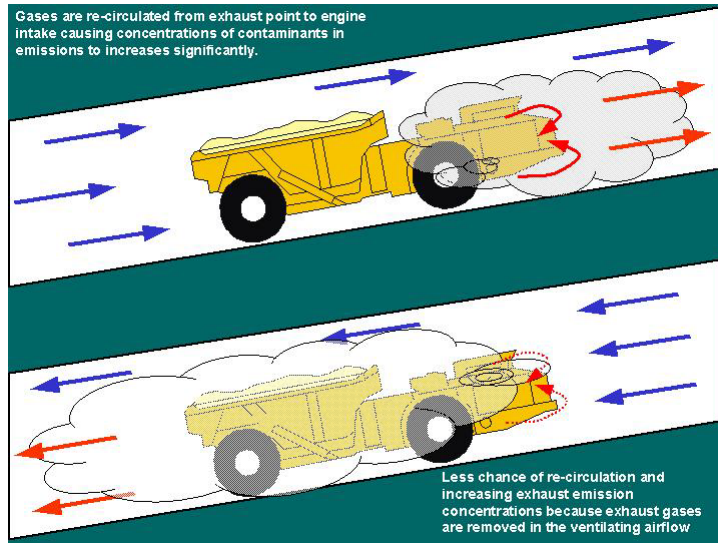
In some cases the airflow in the decline may in fact reverse direction as the truck travels in the drive, usually returning to “the norm” once it has passed. This only occurs in areas with extremely low ventilating pressure and other connections that operate in parallel with the decline. This fluctuation of airflow may also occur in workings connected via a hoisting shaft.

The consequences of any fluctuation cause by vehicles travelling around the mine are seldom great enough to warrant rectification. The main area for concern is when trucks travel up ramps as they do so in an envelope (recirculation) of engine exhaust contaminants. This occurs irrespective of the direction of airflow but when the direction of flow is UP the ramp and the truck is travelling in the same direction there is an increase in the potential for contaminants to build up to levels that will exceed the recommended TWA limits.

For example, if we assume trucks travel up ramps at a speed of 10 kph (2.8 m/s) and the airflow in the $5.5 \text{ m} \times 5.5 \text{ m}$ ramp is $50 \text{ m}^3/\text{s}$ then the air velocity in the ramp is 1.7m/s. This gives a relative velocity between the truck and the air of 4.5 m/s, with the air flowing down the ramp and 1.1 m/s with the air flowing up the ramp. In the second case, it should be noted that the truck is travelling faster than the air.

To achieve a reduction in exhaust emission contamination levels the direction of airflow should be down the ramp. This increases the relative velocity between the air and the truck, which in turn increases the turbulence of air around the vehicle causing the envelope to break up and significantly reduce the level of exhaust emission and dust contamination to which the operator is potentially exposed.

In development headings causing air to flow down the ramp will require the installation of an exhaust overlap ventilation system. This has the disadvantage of providing contaminated airflow to the working face whilst the trucks (or for that matter any other diesel powered equipment) are operating in the ramp.



Because of the relative periods of time spent at the face and the hauling in the ramp, forcing ventilation systems are almost always used.

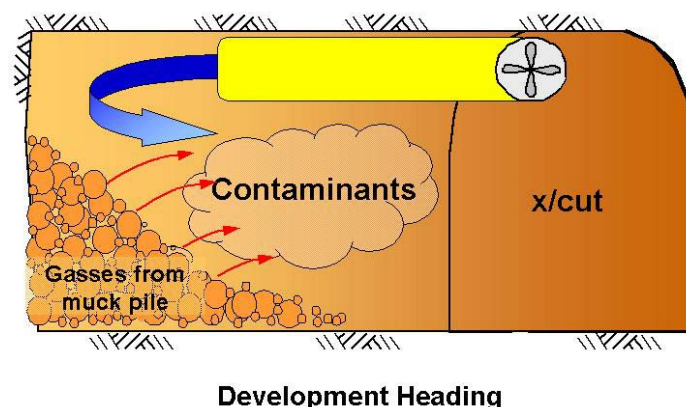
To reduce potential for exhaust emissions to build up to concentrations in excess of the MAC's while travelling up ramps it is vital to ensure that engine performance is matched to the work rate required of the vehicle (i.e. the power to weight ratio). It is also essential to ensure that the engine is tuned and maintained to manufacturers specifications.

7.13 Re-entry after Blasting in a Development Heading

Blasting is an intermittent activity carried out when people are removed from the underground areas affected by the fumes (gases and dust) generated by the blast. The concentration of contaminants in the affected parts of the mine often greatly exceeds the maximum TWA concentrations for a brief period of time until the fumes are diluted by the ventilation currents and removed from the mine. These peaks generally have duration of a few minutes and occur whenever blasting takes place.

Planning and scheduling for rates of advance are usually based upon experience of crews and the normal geotechnical and materials handling requirements of short development headings. For this reason estimates for long heading development rates can and are usually overstated as most neglect to consider the extended re-entry time as a direct result of the length of the heading and the subsequent time required to clear the blasting fumes.

For example consider a long development heading 5.0m x 5.0m x 500m, and an airflow of 25m³/s delivered in a forcing system. The velocity in the drive (Q/A) is 1.0m/s. Assuming that all contaminants are contained in the plug that is being removed and all of the 25m³/s is sweeping the face the minimum clearing time from the face to the primary ventilation circuit, would be only 8.3 minutes and in the scheme of things this is not considered to be of any concern.



Because of duct leakage, duct discharge velocity profiles and the distance of the duct discharge from the face it can be expected that these times could be increased by a minimum of 15 to 25 minutes (i.e. re-entry times from 20 minutes up to 35 minutes).

Gases produced by blasting are to a significant and variable degree either trapped in the blasted muck pile to be released slowly as the muck is removed or dissolved in water naturally present in

the mine or used for dust suppression. It is estimated that up to 60% of the gases produced by blasting are either trapped in the muck pile or removed by dissolution.

Gases produced from blasting include carbon monoxide (CO), carbon dioxide (CO₂), ammonia gases (NH₃), sulphur dioxide (SO₂) and oxides of nitrogen (NO_x). The key contaminants are considered to be CO and NO_x.

Factors that impact on gas production include product manufacture and quality control, the degree of confinement of the explosives (i.e. hole burden and spacing), moisture in the drill holes and the effectiveness of the initiating process.

A wide range of estimates exists of the amount of gas produced per kg of explosives used. In the case of CO references have been found to gas production rates between 1.25 litres per kg and 50 litres per kg. In the case of oxides of nitrogen the range is from 19 litres/kg to 33 litres/kg.

Skochinsky (1969) and De Sousa (1993) have developed algorithms for calculation of gas clearance after blasting in development headings. These algorithms are useful to gain some idea of what to expect but should not be relied on as the sole indicator of when it is safe to return to a blasted area. The only real indicator for safe re-entry is accurate measurement of the contaminants involved.

Skochinsky (1969)

$$t = \frac{7.8}{Q} \times (M_{\text{Explosives}} \times V_{\text{Drive}}^2)^{\frac{1}{3}} \quad \text{Equation 6 Blast Gas Clearing (Skochinsky (1969))}$$

t = Time for the air to clear (seconds)

Q = Quantity of air sweeping the face (m³/s)

M_{Explosives} = Mass of explosives used in the blast (kg)

V_{Drive} = Volume of the excavated drive prior to the blast (m³)

EXAMPLE

Consider a development heading 5.5m high and 5.5m wide 350 metres from the secondary fan. Assuming 200 kg of explosives used in the blast and 20 m³/s is sweeping the face then

Volume of the drive = 5.5 x 5.5 x 350 = 10587.5 m³

$$\text{And } t = \frac{7.8}{20} \times (200 \times 10,587^2)^{\frac{1}{3}}$$

= 1,097 seconds

= 18.2 minutes

deSousa (1993)

$$C_{(x,t)} = \frac{Q}{2A\sqrt{\pi Et}} \times \frac{-(x-vt)^2}{4Et} \quad \text{Equation 7 Blast Gas Clearing (deSousa (1993))}$$

C_(x,t) = Gas concentration (multiply by 10⁶ to get ppm)

t = Time after blast (seconds)

x = Distance to face (m)

Q = Quantity of Contaminant (m³)

A = Cross sectional area of the drive (m²)

V = velocity of the ventilating air (m/s)

E = Dispersion co-efficient (m²/s)

$$E = 28.8 \times v \times Rh \times \sqrt{\frac{\lambda}{\lambda_r}}$$

v = Velocity of the ventilating air (m/s)

Rh = Hydraulic Radius (Area/[height + width]) (m)

λ = Friction Factor of a smooth walled drive. (0.008)

λ_r = Friction factor of the drive (if unknown use 0.015)

This methodology will determine the concentration of the particular gas at a particular time.

EXAMPLE

Consider the development heading above and calculate the gas concentration for NO₂ 19 minutes after the blast.

$$t = 19 \times 60 = 1,140 \text{ (seconds)}$$

$$x = 350 \text{ (m)}$$

$$Q = 0.5147 \text{ (m}^3\text{) from table below}$$

Gas	Gas produced / kg of explosives (kg/kg)	Density of gas (g/m ³)	Volume of gas produced / kg of explosives (m ³)	Mass of explosives used in firing (kg)	Volume of gas produced from firing (m ³)
CO	0.0163	1.25	0.01304	200	2.61
CO ₂	0.1639	1.977	0.082903	200	16.58
NO ₂	0.0035	1.36	0.002574	200	0.5147

$$A = 5.5 \times 5.5 = 30.25 \text{ (m}^2\text{)}$$

$$V = 20/30.25 = 0.66 \text{ (m/s)}$$

E = Dispersion co-efficient (m²/s)

$$E = 28.8 \times 0.66 \times 2.75 \times \sqrt{\frac{0.008}{0.015}} = 38.17 \text{ (m}^2\text{/s)}$$

$$Rh = (30.25/[5.5 + 5.5]) = 2.75 \text{ (m)}$$

$$\lambda = 0.008$$

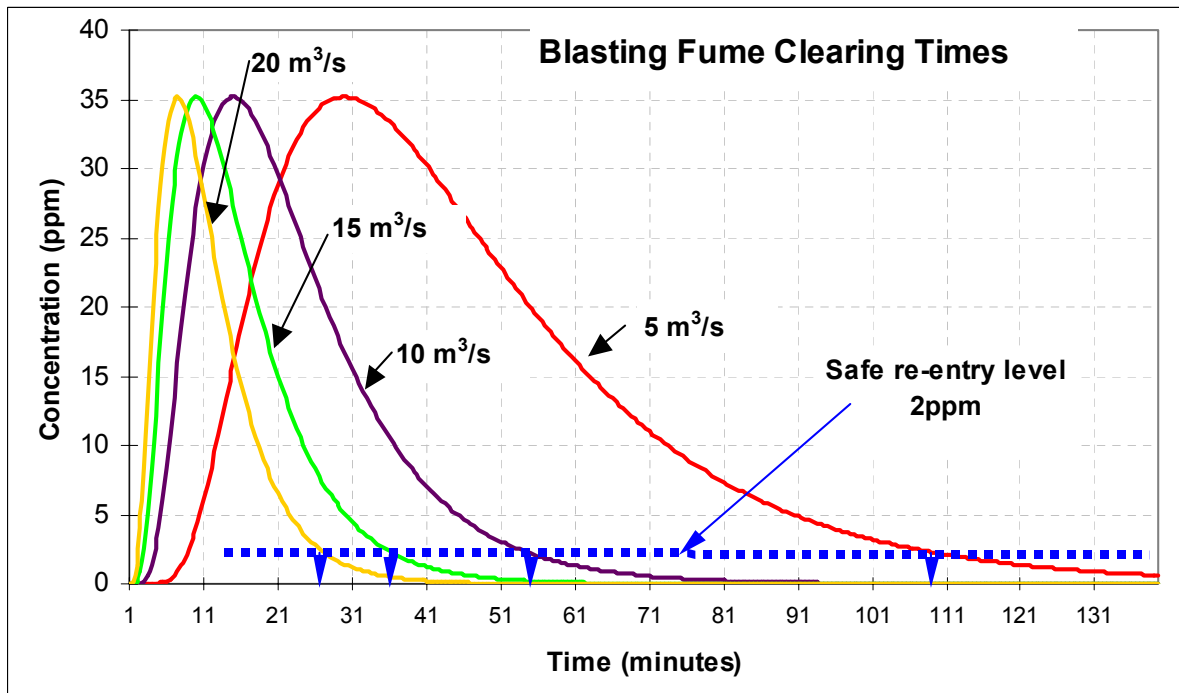
$$\lambda_r = 0.015$$

$$\text{And } C_{(x,t)} = \frac{0.5147 \times \frac{-(350 - 0.66 \times 1,140)^2}{4 \times 38.17 \times 1,140}}{2 \times 30.25 \sqrt{3.142 \times 38.17 \times 1,140}} = 0.000005 \text{ (m}^3\text{) or 5 (ppm)}$$

EXAMPLE 2

This methodology was used to estimate the clearing times for nitrogen dioxide (NO₂) from a development heading 5.5m high and 5.5m wide 350 metres from the secondary fan.

The gas NO_2 has been used because of its high toxicity, low exposure standard and the fact that it will take longer to dilute to safe levels than other gasses produced by blasting. The results are shown graphically below.



This clearly demonstrates the importance of adequate airflow sweeping the workplace.

Howes (1982)¹⁹

The following methodology has been adopted by some Australian mining companies as a standard to establish the earliest re-entry time.

$$t_c = \left(\frac{V}{Q} \right) \times \ln \left(\frac{G_C}{G_R} \right)$$

Where: t = Time to achieve the require gas concentration (seconds)

V = Volume of the gas filled space (m^3/s)

Q = Quantity of air sweeping the face (m^3/s)

G_C = Initial gas concentration (ppm)

G_R = Gas concentration required (ppm)

There are a number of steps required before the use of this equation

1. Calculate the fume throwback distance. (This is the initial volume occupied by the blast fumes)

$$L = \frac{(K \times M)}{F_A \times D \times \sqrt{A}}$$

Where: L = Length of the fume throwback (m)

K = Constant (Assume 25 for development headings)

¹⁹ HOWES. M.J., "Advanced Ventilation Workshop" (1998)

M = Mass of explosives used. (kg)

F_A = Face advance (m)

D = Density of the rock (kg/m³)

A = Area of the face (m²)

2. Calculate the Volume of gas produced (m³)

Gas	(A)	(B)	(C)
	Gas produced / kg of explosives (kg/kg)	Density of gas (g/m ³)	(A) x (B) Volume of gas produced / kg of explosives (m ³)
CO	0.0163	1.25	0.01304
CO ₂	0.1639	1.977	0.082903
NO ₂	0.0035	1.36	0.002574

3. Calculate the concentration of gas in the in the drive. Assume the drive volume after the blast

$$G_{\text{DRIVE}} = \frac{G_{\text{PRODUCED}} (\text{m}^3/\text{kg}) \times \text{Quantity of explosives (kg)}}{\text{Volume of the Drive (m}^3\text{)}} \times 10^6 (\text{ppm})$$

4. Calculate the time taken to mix in the drive. (At this step assume perfect mixing of the gas and the ventilating air)

$$\text{Time for mixing (s)} = \frac{[\text{Length of drive (m)} - \text{Distance of throwback zone (m)}] \times \text{Area of drive (m}^2\text{)}}{\text{Airflow (m}^3/\text{s)}}$$

5. Calculate the time for dilution to the exposure standard using the Blast Clearing

$$\text{Equation above } t_c = \left(\frac{V}{Q} \right) \times \ln \left(\frac{G_c}{G_R} \right)$$

6. Determine the total time for clearing in seconds

$$\text{Clearing Time (s)} = \text{Time for mixing} + \text{Time for dilution}$$

EXAMPLE

Consider a development heading 5.5m high and 5.5m wide 350 metres from the secondary fan. Assuming the face advance is 3.0m, the rock density is 3.1 (kg/tonne), 200 kg of explosives used in the blast and 20 m³/s is sweeping the face.

1. Calculate the fume throwback distance

$$L = \frac{(K \times M)}{F_A \times D \times \sqrt{A}} = 97.75 \text{ (m)}$$

Where: L = 350 (m)

K = 25

M = 200 (kg)

F_A = 3 (m)

D = 3.1 (kg/m³)

A = 30.25 (m²)



2. Calculate the Volume of gas produced (m³/kg)

Gas	(A)	(B)	(C)
	Gas produced / kg of explosives (kg/kg)	Density of gas (g/m ³)	(A) x (B) Volume of gas produced / kg of explosives (m ³)
CO	0.0163	1.25	0.01304
CO ₂	0.1639	1.977	0.082903
NO ₂	0.0035	1.36	0.002574

3. Calculate the concentration in drive

$$G_{\text{DRIVE}} = \frac{0.002574 \times 200}{10542} \times 10^6$$

$$= 49 \text{ ppm}$$

4. Calculate the mixing time

$$\text{Time for mixing (s)} = \frac{[350 - 97.5] \times 30.25}{20}$$

$$= 382 \text{ seconds}$$

$$= 6.4 \text{ minutes}$$

5. Calculate the time taken to dilute the gas.

$$t_c = \left(\frac{10542}{20} \right) \times \ln \left(\frac{49}{2} \right)$$

$$= 2,064 \text{ seconds}$$

$$34.4 \text{ minutes}$$

6. Calculate the time taken for re-entry

$$\text{Time before re-entry} = 6.4 + 24.5 = 30.9 \text{ minutes}$$

SUMMARY OF EXAMPLES

Method	Result (minutes)
Skochinsky (1969)	18.2
De Sousa (1993)	25.9
Howes (1988)	30.9

Re-entry times are best established by measurement of the relevant contaminants combined with the local knowledge and experience of the people directly involved in the blasting and re-entry process.

7.14 Gases from Sulphide Orebodies

See Section 2.2.4 Explosive Dusts page 12

7.15 Gasses from Diesel Engines

See Section 2.5 Diesel Engines page 41

7.16 Control of Mine Gases

7.16.1 Prevention

Preventing the formation of harmful mine gasses is a simple and obvious control method. Examples of this approach include:

Diesel exhaust gas concentrations can be reduced by using high efficiency engines with electronic mixture control or by using alternative power sources (e.g. electric LHD's, shaft hoisting etc).

Ammonia formation can be controlled by reducing ANFO spillage and shot-crete rebound.

7.16.2 Extraction

The aim with this option is to extract gasses via as direct a route as possible to the surface. Examples include:

Uranium mine - ventilation system. Radon daughter laden air is extracted from the stope void and flows directly to the exhaust ventilation system.

Coalmine - gas drainage. Methane drainage boreholes are bored into gassy strata in advance of the face. Gas is extracted from the boreholes.

7.16.3 Isolation

This involves separating personnel from areas where high concentrations of harmful gasses are known to be present. Examples include:

Uranium mine and coalmine return airways.

Mine locations that may be affected by blast fumes at firing time.

7.16.4 Containment

It is sometimes possible to seal off areas (dangerous gas concentrations occur) from the rest of the mine. An example includes:

Bulk-heading off old workings. (i.e. those in which harmful gasses are known to occur). Sealing agents can include shot-crete or urethane coatings. It is normally wise to vent the voids to exhaust, since even with extremely efficient sealing, it is difficult to control leakage due to changes in barometric pressure etc.

7.16.5 Dilution

Dilution of undesirable gasses with fresh air represents a simple and effective method of control. The required airflow rate can be calculated from the following formula:

$$Q = \frac{Q_g}{AC - NC} \quad \text{Equation 8 Dilution Equation}$$

Where:

Q = The required fresh air flow rate (m^3/s)

Q_g = The gas flow rate (m^3/s)

AC = The target gas concentration

NC = The normal concentration of the gas in fresh air

Example:

Methane at 2% concentration has been detected in air flowing out of a stope via a drill drive. The measured flow out of the drill drive is 2 m³/s. How much fresh air needs to be supplied to the drill drive (with a flame-proof fan and anti-static vent duct) to bring the methane concentration down to 1%?

$$\text{Methane flow rate} = \frac{2}{100} \times 2 \text{ m}^3/\text{s} = 0.04 \text{ m}^3/\text{s}$$

The normal concentration (NC) of methane in air is 0.

Substituting...

$$Q = \frac{0.04}{0.01 - 0} = 4 \text{ m}^3/\text{s}$$

i.e. 4m³/s of fresh air is required to dilute the methane to 1% concentration.

7.17 Effect of Atmospheric Changes on Mine Strata Gases

Many underground coalmines have goaf areas (old worked out areas), which receive little or no ventilation. These areas are usually sealed off and this allows the accumulation of gases such as methane. As long as the gases remain within the goaf area there is no safety problem. However, when the atmospheric conditions change and the pressure falls, the pressure on the gas decreases and therefore the volume of the gas increases (Boyle's Law). Because the volume of the goaf cannot change some of the gas will be forced out into the ventilation system.

Designing the ventilation system to prevent this must include;

- (a) Sufficient airflow to dilute the gases to harmless mixtures
- (b) Directing them away from men and machinery.

The quantity of air will depend on:

- (a) The total volume of the goaf. The greater the volume of the goaf the more gas will be forced into the ventilation system,
- (b) The rate of fall of the atmospheric pressure. The actual size of the fall is less important than the rate at which it occurs, since the pressure drop determine the rate of expansion of the gas.

The best way to ensure the gas is directed away from the work areas is to ensure that the goaf is connected directly to the exhaust air system.

It is important to note that even if the goaf is sealed off they will nearly always "breath".

7.18 Spontaneous Combustion

Although coal is considered to be a commonplace commodity it is extremely complex and differs widely from seam to seam. Certain coal seams and carbonaceous shales have a tendency to oxidise at normal working temperatures and others are prone to spontaneous combustion. A great deal of work has taken place over many years in an attempt to identify the exact components of coal that cause self-heating but as yet it is still not fully understood.

What is known is seams liable to spontaneous combustion are those,

- thick or a composite,
- are of inferior quality,

- are low rank (i.e. relatively young),
- contain pyrites,
- readily adsorb and desorb moisture when there is a large gap in humidity between the coal and the air, and
- have a high moisture holding capacity

The spontaneous combustion process starts with oxidation at normal temperature (say 20°C) and the rate of oxidation is dependant upon

- the temperature,
- percentage of oxygen and,
- humidity

If the oxygen used up in the oxidation process is not replenished the rate of oxidation will slow and eventually cease when the content fall below 2%.

The heat produced by the oxidation process causes a rise in temperature, increasing the rate of oxidation and again increasing the temperature of the coal. The presence of fine coal increases the rate of oxidation due to the relatively large surface area exposed to the ventilating air. Once the temperature reaches 80 - 100°C the oxidation process rapidly increases until the temperature reaches 650°C and ignition occurs. If sufficient heat is removed in the ventilating air then the heating will slow and temperatures will remain below ignition temperatures.

The classic spontaneous combustion occurs in broken 'dry' coal of mixed size with sufficient fines to have a large surface exposure and large lumps that create the voids for the passage of humid air. The airflow would be in sufficient quantities to replenish the oxygen but insufficient to remove the heat.

Spontaneous combustion is most likely the result of poor mining practices such as:

- leakage of air through the goaf (this could be the result of high ventilation pressures) or,
- a poor extraction sequence that leaves large stocks of broken coal.

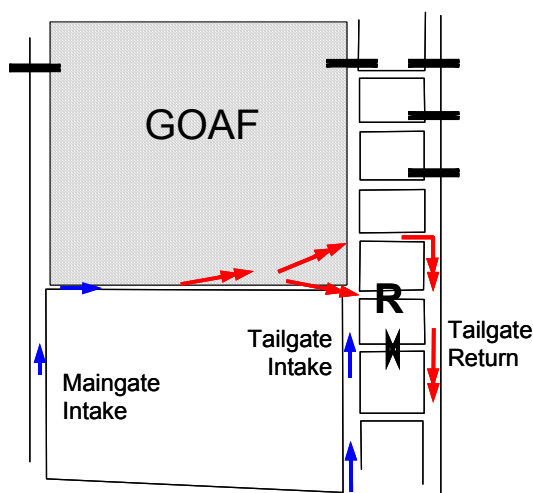
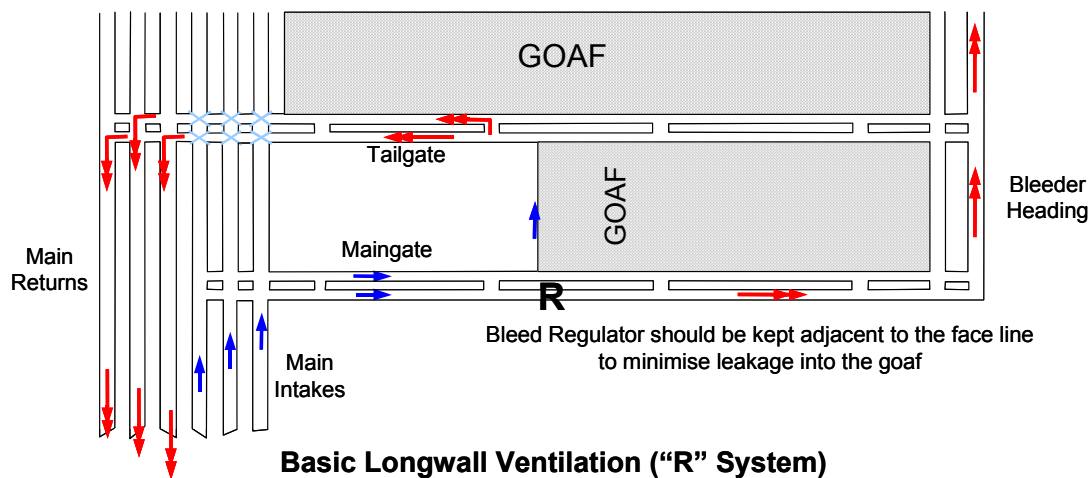
There are a number of signs that indicate a heating is in progress.

1. **'Sweating'** is the condensation of moisture on the roof, ribs or metal straps and washers used in ground support. This results from high temperatures in the goaf area driving out the moisture contained in the coal. This hot now humid air condenses on the cooler surfaces outside the goaf.
2. **'Haze'** or **'fog'** caused when the hot humid air from the goaf condenses in the cooler ventilating airflow.
3. A **'smell'** comes under many names such as 'goaf stink', 'fire stink' or 'stink damp'.
4. **'Hot air'** from the goaf is an indication that a heating is well advanced.
5. **'Smoke'** a fire is imminent.

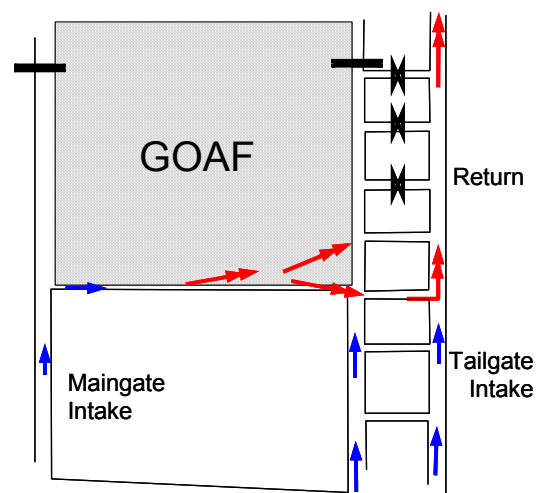
In a longwall retreat goaf there is a region immediately behind the face in which the quantity of air is sufficient to remove the heat of oxidation. Deep within the goaf the quantity of air is insufficient to allow the oxidation process to continue and in fact the oxidation process uses up the oxygen. Between these regions there is a region that is conducive for spontaneous combustion. As the longwall face retreats this region also retreats and remains at the same distance behind the face.

The circuits commonly used in Australia to prevent spontaneous combustion are shown below.

Bleeder headings must be adopted with extreme caution as they pose a risk of promoting a heating or conversely having an unventilated goaf allowing a build up of gases that could be forced out by a change in atmospheric pressure.



Back Return System



"Y" Return System

7.19 Control of Dust

Once dust is liberated from the rock, it is transported to various parts of the mine in airborne form via the mine airways. When the relative velocity between a rock surface and the ventilating air exceeds 5.0 m/s the inspirable dust fraction adhered to the rock surface may be dislodged and will become entrained into the air stream. In underground mines this may occur whenever rock is moved from one point to another. The dust may come from two sources the exposed surface of the rock being transported or, from the roadway surface when dust, picked up by the tyres of vehicles, is entrained into the airstream. These occurrences are usually overcome by wetting the rock on either the load being transported or the roadway surface. Obviously in roadways with air velocities less than 5.0 m/s the problems associated with dust from tyre pick up is much reduced.

It is normal practice to apply water from either a water cart or overhead sprays onto underground roadways to allay dust pick up from tyres. The consequences of these systems are either not enough water or some times excessive water, particularly from overhead sprays, that causes erosion of the roadway that in turn requires constant maintenance to the roadway surface. In open-pit mining the use of additives to water has been used in attempts to reduce the roadway maintenance requirements. In recent years a number of underground mines have trailed these additives with varying degrees of success.

Dust suppression agents come generally as water additives and are described by their respective manufacturers as either,

- Soil wetting agents (detergents),
- Binding agents (surfactants), or
- Compaction enhancers.

Most of these agents have been manufactured with the objective to reduce roadway surfaces breaking up therefore reducing maintenance costs. It is important to note that all products are susceptible to water and have the same requirements for well constructed roadways, sloped to prevent water pooling. Some binding agents cause surfaces to become very slippery and any use of these products must be carefully considered.

Water is the main cause of roadway erosion and poorly constructed roadways will always have high maintenance costs irrespective of the use of any wetting or binding agent for dust suppression.

The three main methods for controlling dust in underground mines include:

- Application of water
- Dilution Ventilation
- Localised extraction ventilation

Water. Application of water is by far the most popular (and usually the most practical) dust control method. Wetting the dust prevents it from becoming airborne. To be effective, the water should be applied (by spray jets) to the point at which the dust is liberated (e.g. at the drill bit, on the roadway, at the drawpoint etc). It is important to try to ensure that the spray water is as clean as possible. Muddy re-cycled mine water may introduce more dust into the mine when the water evaporates. Some dust types may be difficult to wet effectively with water and wetting agents called surfactants may need to be added to the water. A rule of thumb design criterion is to aim to supply between 1 and 5% of water to broken rock (on a weight for weight basis). It is not usually considered safe or advisable to apply water at ore handling facilities. This is because of the serious danger of mud rushes occurring in vertical storage facilities if an excessive build-up of water occurs.

Dilution. In most underground mines, dust is produced at diverse and often numerous locations. The amounts of dust produced at the individual locations are sometimes quite small. In these circumstances, the reduction of dust concentration by dilution with an appropriate airflow rate is a legitimate method of dealing with airborne dust. Determining the amount of airflow required to adequately dilute dust on a mine wide basis is surprisingly difficult. The determination is complicated by the cyclic and dynamic nature of the mining process, the effectiveness of watering down dust and many other factors. Some design factors, which may assist, are listed below, however it must be emphasised that no two mines are the same and application of the factors must be tempered with practical experience:

- In many mines, it has been found that satisfactory dust levels can be maintained if sufficient air is circulated to satisfy diesel exhaust dilution criteria (e.g. 0.05 m³/s per kW of rated diesel engine power).
- In all cases, the minimum air velocity to ensure adequate mixing and dilution must be greater than the lower limit of turbulent flow (about 0.1 m/s).
- Air velocities in intake airways and in airways in which men travel should be less than about 6 m/s. At higher velocities, increasing problems with re-entrainment of settled dust will be experienced.

7.19.1 Dust Extraction

This involves exhausting dust-laden air from a point as close as possible to where the dust is generated, so that dust can be prevented from entering the main intake mine airways. This type of system is best suited to fixed facilities, such as ore passes, crushers and conveyor belt transfer points. In underground mines, it is almost always preferable from an overall cost (capital + operating) viewpoint to exhaust the dusty air directly to an exhaust airway. There may be some circumstances where the dust may need to be filtered so that the extracted air can be re-used. The most suitable filtration systems for the underground environment are generally wet type dust scrubbers, although fabric bag-house type filtration is sometimes also used.

Dust is a result of the disintegration of matter and the size of the dust particle produced is determined by the impact per unit area. For example striking a rock with a hammer will split the rock into large pieces forming coarse dust particles. If we were to use the same force using a chisel it would break only a small piece of the rock into fine dust particles because the force is directed onto a much smaller area.

Disintegration processes occur in many ways with crushing, grinding, blasting and drilling being the obvious ones. Other processes that cause dust particles to become airborne are those involving the transportation of previously broken material by loaders, uncovered trucks and conveyor belts. The liberation of dust particles to the atmosphere occurs when the velocity of the air relative to the vehicle is sufficiently high enough to cause dust pick up.

	Particle Size(μm)	Pick Up Velocities (m/s)	
		Quartz	Coal
Dry	75 to 100	6.3	5.3
	35 to 75	5.3	4.2
	10 to 35	3.1	3.2
Semi Dry	75 to 100	7.4	6.3
	35 to 75	6.3	5.3
	10 to 35	4.2	4.2

Dust is also liberated to the atmosphere whenever broken material is transferred e.g. conveyor, loading and unloading processes.

The initial step in the design of a ventilation extraction system requires an accurate assessment of the volume flow required to effectively remove the hazard. If this calculation is incorrect then all following decisions regarding the ducting, fans and filters are also likely to be incorrect.

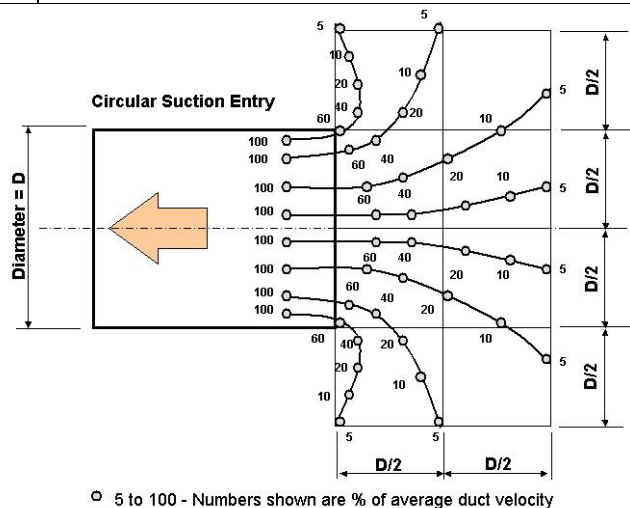
Because a suction exhaust type system requires most of the principles of ventilation to be applied (also that the laws of airflow apply equally to both pressure and exhaust systems) we will focus our attention on the design considerations for an exhausting system.

Ensuring adequate 'capture' of the contaminant is the first consideration and must take into account the following variables of the contaminant to be handled.

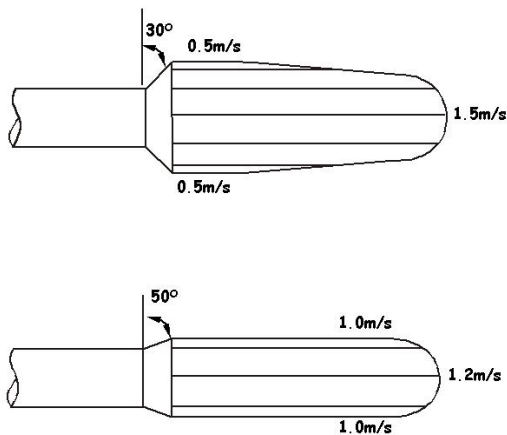
VELOCITIES FOR CAPTURE HOODS

Type Of Emission	Capture Velocity	Example
Lazy Emission	0.25 to 0.5 m/sec	Fume Control
Low Velocity Emissions points	0.5 to 1.0 m/sec	Manual Activities e.g. Aspiration
Active Emissions mechanical screens	1.0 to 2.5 m/sec	Low speed Emissions e.g. Transfer Points, Vibratory Screens.
High Velocity	1.0 to 10.0 m/sec	High Speed and rotary emissions e.g. Transfer Points, Vibratory Screens e.g. Sawing, Grinding, Polishing

Wherever possible capture hoods should be designed to enclose the entire process source. If this is not possible the hood should be located as close to the source as is practical and sited to make use of the direction of flow of the particles caused by the process. This site must be chosen to take account not only of the thermal properties of the process but also the effects of gravity so as to avoid the larger particles falling out before reaching the influence of the hood. It is also important to site the hood in a position that will not allow personnel to come between it and the process source.



7.19.1.1 Hood Design

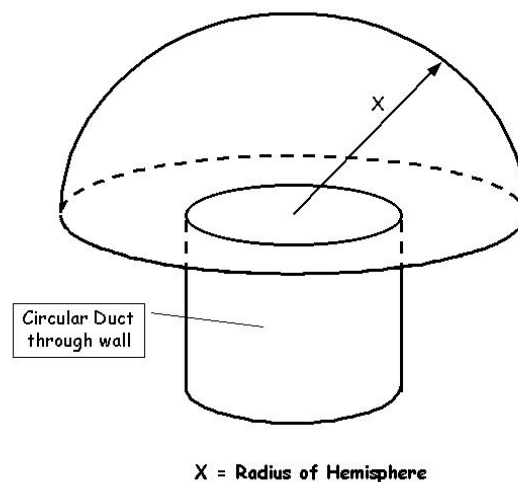


Having determined the capture velocity and angle of the hood the next step is the siting of the hood and unfortunately the economic and engineering importance of this procedure is usually ignored or at least misunderstood.

As shown in the figure below, if we were to hold a piece of string of length X at the centre of the duct and moved

Each hood is designed to control the velocity in all directions at a given distance from a point of exhaust. This is illustrated in the figure opposite that shows the rapid velocity decrease in the vicinity of an exhaust intake.

The efficiency of exhaust hoods can be greatly improved if they are shaped to gradually merge into the ducting and thereby improving the aerodynamics of the airflow.



it in any direction we would circumscribe a semi-circle and finally a hemisphere of radius X. To maintain the capture velocity at any point on the surface of this hemisphere we must ensure that sufficient air is drawn through the duct.

The airflow required to maintain this velocity may be calculated by the equation:

$$Q = VA^3$$

Where

Q = Exhaust airflow quantity (m³/sec)

V = Required capture velocity (m/sec)

A = Area of the geometric shape (m²)

Dalla Valle in the 1930's refined this equation to cater for circular or rectangular hood faces:

$$Q = V (10 \times X^2 + A)^3$$

Where

Q = Exhaust airflow quantity (m³/sec)

V = Required capture velocity (m/sec)

X = The distance from the hood face to the process source (m)

A = Area of the hood face (m²)

However this equation also has its limitations as shown by Fletcher in the '80's and should not be used for rectangular hood face areas with an aspect ratio not equal to 1 i.e. they should only be used for hood face areas that are either square or circular.

$$\text{Aspect ratio} = \frac{\text{Width}}{\text{Length}}$$

Example

What quantity of airflow is required when a hood of 400 mm diameter is placed 320 mm from a dust source with a capture velocity of 3.0 m/sec

$$\begin{aligned} Q &= V(10 \times X^2 + A) \\ &= 3([10 \times 0.32^2] + [3.142 \times 0.2^2]) \\ &= 3.45 \text{ (m}^3\text{/sec)} \end{aligned}$$

If this same dust hood was repositioned 200 mm from the dust source and the capture velocity remains unchanged then the air volume becomes:

$$\begin{aligned} Q &= V(10 \times X^2 + A) \\ &= 3([10 \times 0.2^2] + [3.142 \times 0.2^2]) \\ &= 1.58 \text{ (m}^3\text{/sec)} \end{aligned}$$

These results shows that by moving the hood closer the volume of air required is halved, and the cost of the installation is reduced as a smaller fan will be required.

In general hoods should be designed to approximately cover the area (larger never smaller) of the contaminant source and if this source process is hot, the velocity pressure must exceed the thermally induced draught.

Although we have shown that the closer the hood is to the source, it must never hinder the process otherwise the operator may remove it.

Finally the hood face should be designed to have the smallest possible area that will be acceptable to the process operator.

7.19.1.2 Duct Design

After having decided the capture velocities and calculated the airflow requirements for each hood the next step is to determine the optimum transport velocity inside the duct. This velocity should be such to prevent any settling out of the particles as they are transported through the system. It should be noted at this point that the transport velocities should be slightly higher in main ducts than in branch ducts.

Air Velocities for Dust Transport

Type of Dust (or vapour)	Branch Duct Velocity (m/s)
Paint fumes	8 to 10
Limestone	13
Metal fumes (welding)	15
Sandblasting	18 to 22
Coal	20
Lead dust	28

Either making use of system damper plates or changing the cross-sectional area of the duct can achieve control of velocity inside ducts. The use of damper plates is very expensive as far as system pressure losses are concerned and should be avoided wherever possible, if however it is intended to expand either the system or the process they must be considered. The other occasion they are used is when the installation space available prevents the use of the desired duct configuration.

Duct material, shape, size, branching and bends are all critical as they all have the potential to provide excessive resistance (pressure loss) and add to the size of the fan. To keep these pressure losses to a minimum ducting should be streamlined as far as possible by avoiding sharp bends or sudden changes of cross-section or cross-sectional area. Branch pipe sizes must then be calculated to provide the volume of air required at each point.

Duct design should provide for maintenance after the plant has been installed and should therefore provide sufficient inspection and cleaning openings. It is also important to take precautions against corrosion and abrasion as particles may cause rapid wear in ducts, particularly bends and it is therefore necessary to provide protection at these points.

7.20 Economics of Airflow

The objective of ventilation economics does not differ from the principles of any investment i.e. to receive the maximum return on the investment.

Ventilation circuits are designed to fit the operational requirements for a specific mine. Any economic evaluation is carried out to determine the most cost effective alternative to meet these requirements.

Alternatives evaluated may not necessarily include the optimum economic situation. Many times, it is a compromise between the engineering and the scheduling requirements to determine the most cost effective alternative.

Ventilation costs should be regularly monitored as changing cost in any of the variables may alter the cost effectiveness of circuit design.

It is extremely difficult to provide a single set of economic factors for ventilation studies as costs such as development and power will vary from mine to mine (due to location of the mine from services), rock type, ground conditions, accounting procedures of the mine etc.

7.20.1 Cost of Airflow

If the cost per unit of power to operate the fans is known and the airflow and pressure losses in an airway can be determined then it is possible to calculate the cost of power to provide the airflow.

Example

If the cost of electrical is \$0.10/kW hour, what would be the annual cost of power for a flow of 150m³/s in a 3.0m diameter 200m long concrete lined airway?

From Atkinson's ventilation equation

$$P = \frac{KCL}{A^3} \times Q^2$$

then determine the pressure losses for the airway (for a concrete line airway assume $k = 0.004 \text{ N s}^2/\text{m}^4$)

$$\begin{aligned} &= \frac{0.004 \times 9.425 \times 200}{7.064^3} \times 150^2 \\ &= 481 \text{ Pa} \end{aligned}$$

the airpower is

$$\begin{aligned} \text{AP} &= P \times Q \\ &= 481 \times 150 \\ &= 72,150 \text{ watts} \\ &= 72 \text{ kW} \end{aligned}$$

and the cost of power is \$0.10/kW per hour then

$$\begin{aligned} \text{Cost of Power} &= \text{AP} \times \text{unit cost of power} \\ &= 72 \times \$0.10 \\ &= \$7.20/\text{hour} \\ \text{or} &= \$63,072/\text{year} \end{aligned}$$

If the diameter of the shaft was increased to 4.0m then the cost of power would be:

$$\begin{aligned} P &= \frac{kCL}{A^3} \times Q^2 \\ P &= \frac{0.004 \times 12.566 \times 200}{12.566^3} \times 150^2 \\ &= 114 \text{ Pa} \end{aligned}$$

and Air Power ($P \times Q$) = 114 x 150

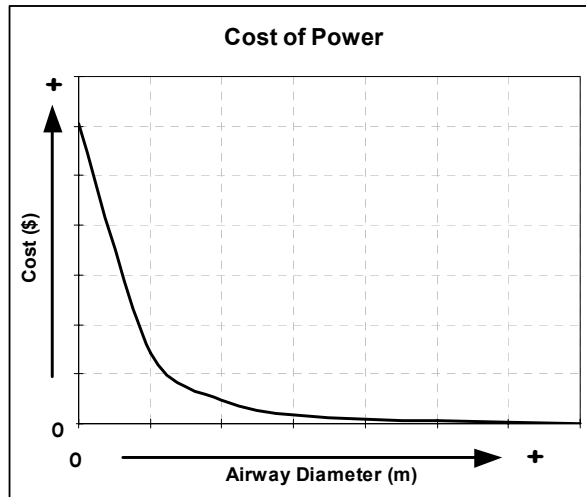
=1710 Watts

or = 1.710 kW.

and cost of power for 1 year would be:

= $17.1 \times 0.10 \times 24 \times 365$

= \$14,979



From this it becomes obvious that the cost of power decreases as the cross sectional area of the airway increases. Although the cost of power initially decreases quite rapidly it then levels out until the cost savings become relatively insignificant with each incremental increase in airway cross sectional area.

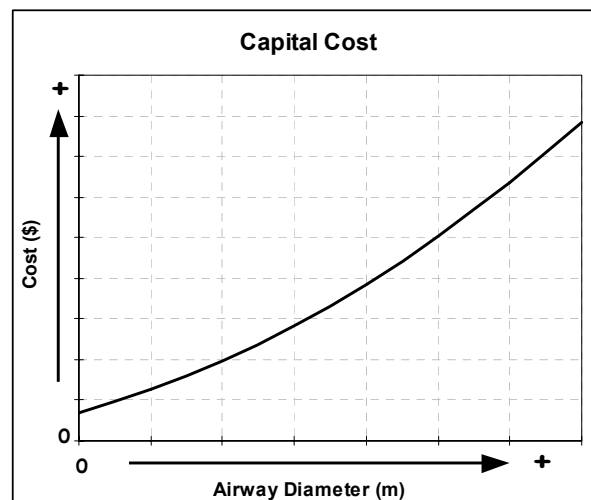
7.20.2 Optimum Airway

In most mines, the airways of highest resistance are the primary intake and exhaust airways. The pressure losses in these airways can be 90% of the total mine losses and therefore require most consideration.

Besides the operating costs of power there is also the capital cost for development of the airway. Simplistically the larger the cross sectional area of the airway, the more rock to be broken and removed therefore the higher the cost. I.e. the cost for development increase as the cross sectional area of the airway increases.

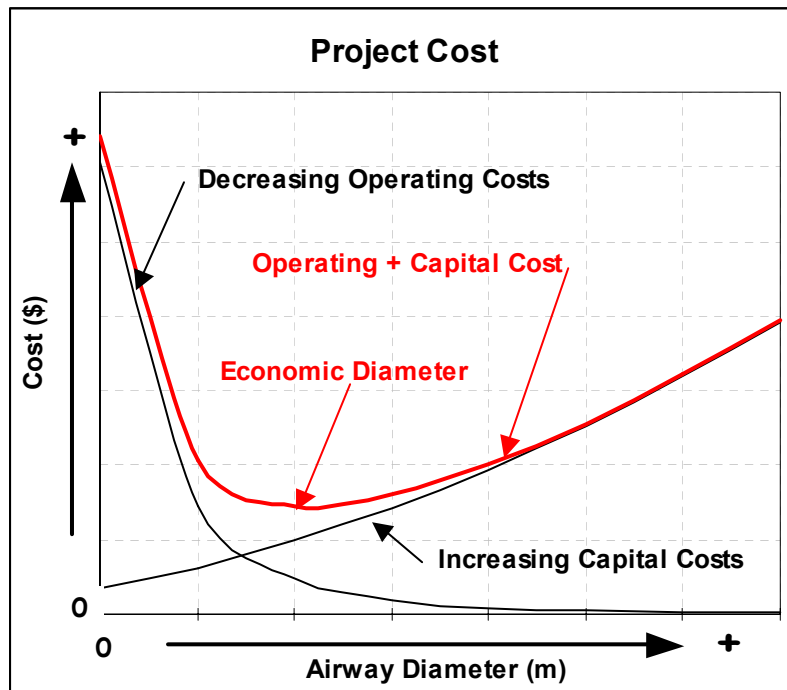
One of the frequently used terms associated with ventilation economics is "optimum diameter", "optimum velocity" or "optimum cross-sectional area".

Using the variable operating (power) cost and fixed capital (development and fans) the airway can be evaluated to determine the lowest total cost for a specific quantity of air over the life of the airway.



This is achieved by the simple addition of the operating costs (power) and the capital costs (development and fans) for a number of airway diameters.

In all cases it is simply that point where the total cost over the life of the airway (or system) is at it's lowest.



7.20.3 Time Value of Money

When considering the operating costs particularly over a long period of time it is important to consider the time value of money. For example if you were to invest \$1,000 at 10% it would be worth \$2,593.75 in 10 years time (excluding tax) and if you wanted to have \$1,000 in 10 years time you would have to invest \$385.54 at 10% today i.e. the Present Value of \$1,000 in 10 years discounted at 10% is \$385.54.

This concept of Present Value makes no mention of inflation. This is because inflation influences the purchasing power of the money not the amount of money.

The Time Value of money refers to the effect that interest rates (as returns from potential investment opportunities) can have on the value of money and not the effect of inflation.

When considering expenditure on ventilation airways, we should consider that the money need not be spent and it could be invested. Thus for a given project, the expenditure must return the original and operating expenditure equivalent to interest rates which could be earned by investing the money.

The future sum of our investment can be calculated from

$$F = P(1+i)^n$$

Where

F = Future Value (\$)

P = Present Value invested (\$)

i = Interest rate % (rate of return on capital)

n = Number of interest compounding periods usually yearly

Example

What is the future value (F) of \$1,000 at the end of 6 years if interest is 10%?

$$\begin{aligned} F &= P(1+i)^n \\ &= 1,000(1+0.1)^6 \end{aligned}$$

$$= \$1,771.56$$

Similarly, if we know the value of a future sum we can determine the Present Value by solving for P. i.e.

$$\begin{aligned} P &= \frac{F}{(1+i)^n} \\ &= \frac{1,000}{(1+0.20)^{20}} \\ &= \$26.08 \end{aligned}$$

This shows that \$26.08 invested today for 20 years at 20% interest will grow to \$1,000.

Both of these functions deal with single payment present values.

If we were to invest money as a series of equal investments, each investment draws compound interest for a different number of periods.

The future amount of money from this type of investment is calculated by:

$$F = A \frac{(1+i)^n - 1}{i}$$

Where A = Amount Invested (\$)

Example

Calculate the future value (F) 6 years from now with yearly investments of \$1,000 at 10% interest.

$$\begin{aligned} &= 1,000 \times \frac{(1+0.1)^6 - 1}{0.1} \\ &= \$77,156.61 \end{aligned}$$

This shows the future value of yearly investments of equal value.

Should we wish to know the present value (P) of a series of \$1,000 payments made at the end of each year of 6 years if the compound interest is 10% per year. We can use the following equation.

$$\begin{aligned} P &= \frac{A(1+i)^n - 1}{i(1+i)} \\ &= 1,000 \times \frac{(1+0.1)^6 - 1}{0.1(1+0.1)^n} \\ &= \$4,355.26 \end{aligned}$$

Thus the Present Value of a series of \$1,000 payments made each year for 6 years with a compound interest of 10% \$4,355.26.

7.21 Rule-of-Thumb Principles and Design Factors

There are a large number of ventilation rule-of-thumb (ROT) principles that have evolved over many years, many are contradictory and many are now no longer applicable. When using any

ROT they must be placed in the correct context and this can only be done successfully if there is a good understanding of the basis from which the specific ROT has been derived.

Although experience should not be ignored and basing ventilation design specification on past and present practices in comparable mines can and has led to near disastrous consequences of over and under ventilated mines.

There is some basis for the use of ROT principles particularly when attempting to “get a feel” for what may be required, but mine ventilation systems must be designed specifically for each and every mine, as no two mines are identical.

One site on the Internet (www.mcintoshengineering.com) provides a comprehensive list of ROT principles. Before using any of these values the origin and circumstances of the factor should be fully understood.

Listed below are some factors encountered by the author over many years.

	Rule-of-thumb factor	Comment
(a)	Fresh air quantities 0.024 – 0.094 m ³ /s per tonne of ore and waste produced in the mine (it is assumed that this refers to a daily production rate)	Consider a modern mechanised mine producing 1,000,000 tonne of ore and waste. From (a) the quantity of air should lie between 65 and 257 m ³ /s
(b)	0.094 – 0.47 m ³ /s per man underground	A modern mine may have 50 people underground therefore from (b) the quantity of air required is between 4.7 and 23.5 m ³ /s. However in a labour intensive operation there could be 2,000 people underground and the airflow would be between 188 and 940 m ³ /s.
(c)	0.063 – 0.126 m ³ /s per kW of operating diesel equipment.	For diesel equipment 4,000 kW will require 252 and 504 m ³ /s
	Velocities in airways Service Shaft 5.1 – 7.6 Production Shaft 7.6 – 10.2 Main Entry 2.6 - 7.6 Conveyor tunnels, declines 2.5 – 5.1 Inlet and exhaust rises (no access) ±10.0 Inlet and exhaust rises (access required) 2.5 – 5.1 Exhaust mains 10.2 – 15.2 Exhaust shafts (concrete lined) 15.2 - 20.3 Exhaust shafts (rock section) 10.2 - 15.2 Avoid 7.5 to 12.5 m/s in saturated airways.	

“in today’s increasingly uncertain, competitive and fast moving world, companies must rely more and more on individuals to come up with new ideas, to develop creative responses and push for changes before opportunities disappear or minor irritants turn into catastrophes.

Innovations, whether in products, market strategies, technological processes or work practices, are designed not by machines but by people.”

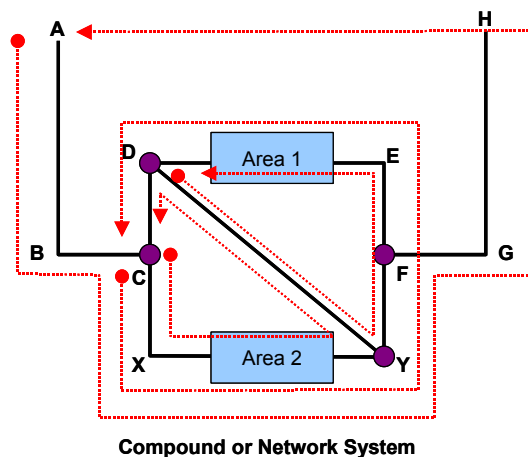
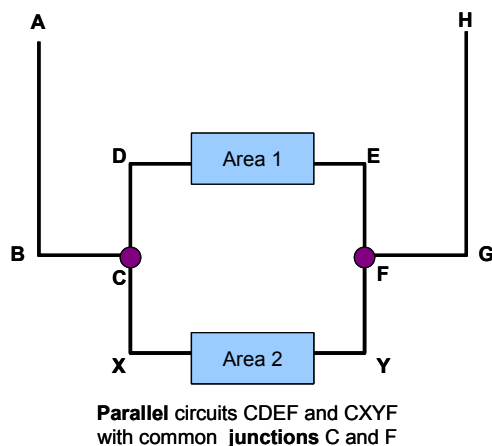
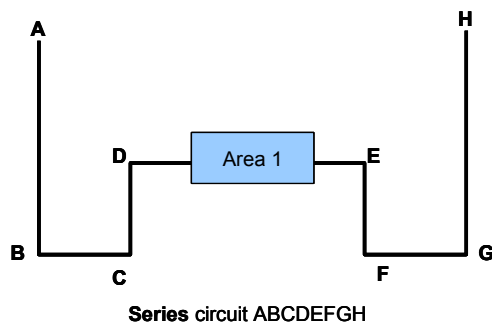
Rosabeth Moss Kanter - “The Change Masters”

8 NETWORKS AND COMPUTER MODELLING

8.1 Ventilation Networks

In section 4 we looked at simple series and parallel circuits. Mines consist of many airways (including stope voids open orepasses etc.) interconnected in such a way that it is impossible to reduce them to simple series or parallel circuits.

This network of airways can, with the use of computers be analysed to predict airflow quantities and pressures. Consider the following figure to represent three possible stages for development of a mine. At first a simple series circuit, expanding when a new area is opened up creating a parallel circuit and finally the connection of the two working areas resulting in a network of airways.



The construction of airway DY now presents a different problem consisting of branches, junctions and meshes.

Branches (connect junctions) and are ABC, CD, DEF, DY, CXY, YF and FGH. (Total 6)

Note: a branch is any series of airways and may have different dimensions and must be included at least once in the group of meshes

Junctions (two or more branches)
C, D, Y and F. (Total 4)

Meshes (and closed circuit)
CXYDC, DYFED, CXYFEDC and ABCXYGHA. (Total 4)

The solution of the series and parallel was explained in section 4. The solution of networks is undertaken with the following assumptions:

1. Air is incompressible and,
2. The airflow in the network obeys Atkinson's law

Kirchhoff's electrical laws can be restated and applied to ventilation networks i.e.

1. The total quantity of air entering a junction is equal to the total quantity leaving the junction.

$$\sum_{i=1}^S Q_i = 0$$

2. The sum of the pressure changes in a mesh is zero.

$$\sum_{i=1}^m (P_i - P_{fi}) - (NVP)_m = 0$$

Where density is constant

m = number of branches forming a closed mesh

P_i = frictional pressure drops.

P_{fi} = total pressure drop across the fan

NVP = natural ventilating pressure.

In the simplest case of a mesh without a fan and no NVP this reduces to

$$\sum_{i=1}^m P_i = 0$$

These equations are only correct when all terms are considered at standard density

8.2 Analysis of a Network²⁰

The solution of a network requires the determination of the characteristic of each and every airway, the number of branches, the number of junctions and the number of meshes.

From Kirchhoff's 1st law for a network containing 'b' branches and 'j' junctions there will be 'b' airflows to determine and therefore 'b' equations must be solved.

Similarly from Kirchhoff's 2nd law there will be 'j' equations to be solved. However only $j - 1$ of these are independent because the j^{th} junctions are already defined by flows at other junctions and therefore there are $b - (j - 1)$ equations to construct. These are obtained by choosing a minimum of, $b - j + 1$ meshes

and writing equation. $\sum_{i=1}^m (P_i - P_{fi}) - (NVP)_m = 0$ for each of them. This gives $j - 1$ junction equations from

Kirchhoff's 1st law and $b - j + 1$ mesh equations from Kirchhoff's 2nd law.

When written in terms of $P = RQ^2$ a quadratic equation for each mesh can be developed to find the solution. For a network containing m meshes the powers of Q would be 2^{m-1}

²⁰ For a detailed discussion on Network analysis refer to McPHERSON, M.J., "Ventilation Network Analysis" Environmental Engineering in South African Mines. (1989) Chapter 8. pp211 - 239

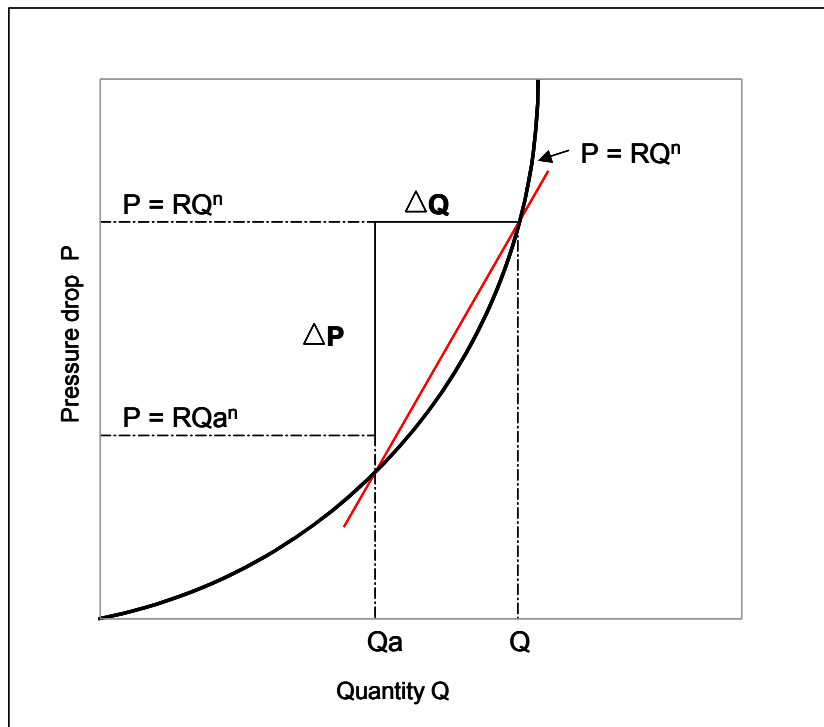
In relatively simple networks there is an application of this method but it falls short in large networks in much the same way as simple series and parallel circuits in networks, and other techniques should be considered.

8.2.1 Hardy Cross

The technique of fluid network analysis involves making an initial estimate of the flow distribution and calculating an approximate correction to be applied to the flow in each branch and then iterating the correction until an acceptable result has been achieved.

The technique that has found its place in mine ventilation calculations is the one developed by Professor Hardy Cross in 1936.

Consider the figure below and assume a quantity Q passing through a duct.



n is the constant for the range of flow considered. To determine Q an estimate of Q_a is made i.e. $Q = Q_a + \Delta Q$ where ΔQ is the error for quantity. Similarly ΔP is the corresponding error for pressure.

Differentiating $P = RQ^n$ we find

$$\text{Slope of the curve } \frac{\Delta P}{\Delta Q} = nRQ^{n-1}$$

Or at point P_a

$$\frac{\Delta P}{\Delta Q} = nRQ_a^{n-1}$$

$$\Delta Q = \frac{\Delta P}{nRQ_a^{n-1}}$$

But $\Delta P = RQ^n - RQ_a^n$

Therefore $\Delta Q = \frac{RQ_n - RQ_a^n}{nRQ_a^{n-1}}$ which is the “out of balance” equation in a single duct.

Considering a network with b branches and applying Kirchhoff's laws and assuming the index $n = 2$ for a fully turbulent flow we can derive the mesh correction factor Q_m to be

$$\Delta Q_m = \frac{-\left\{ \sum_{i=1}^b R_i Q_{ia} |Q_{ia}| - P_{fi} \right\}}{\sum_{i=1}^b (2R_i |Q_{ia}| - S_{fi})}$$

$|Q_{ia}|$ is the absolute value and S is the slope of the fan characteristic.

The procedure for the use of this method is described as follows:

- (a) Estimate the quantity of air flow through each branch
- (b) Decide on a pattern of closed meshes and determine the number. The minimum number is determined by
No. of branches – No. of junctions + 1
- (c) For each mesh evaluate the mesh correction factor ΔQ_m .
- (d) Correct the flow in each branch.
- (e) Repeat steps (c) and (d) until values of ΔQ_m are below the prescribed value.
- (f) Repeat steps (b) to (e) for each of a number of changes to the network.

8.3 Computer Modelling

Modelling of ventilation systems with computers can be an excellent tool to assist with ventilation system design. However, ventilation simulations are not a magic panacea. They do not replace a good working knowledge of basic ventilation theory and practice and the following problems are very common:

- The “user friendliness” of the latest software and the ability to directly import 3D mine design data has encouraged large numbers of inexperienced, but computer literate engineers to have a go. Impressive graphical output is produced, but the answers are often wrong.
- Inexperienced users often use the default input values (because the user is often unsure what the proper values should be). In one popular computer package, the supplied shock loss equivalent lengths are grossly misleading and should never be used.
- Some computer packages add shock loss factors automatically (by default) to every mine airway. This can cause significant problems, particularly if the model contains a large number of short airways.
- The models are often not checked against “reality”. This appears to be due to a basic unawareness of how to take simple ventilation pressure measurements and compare these with simulated values.

The aim of this section is to provide some practical advice so that some of these problems can be avoided.

The user of computer simulation software should have a very clear idea of what questions need to be answered – before any data is entered. The main reasons for undertaking a simulation are listed below:

- Estimation of fan duty requirements (for primary fans and circuit fans).
- Estimation of general airflow distribution, in more complex ventilation circuits.

It should be appreciated that computer simulations are not a ventilation design “black box” There are a number of things, which they won’t do, including:

- Standard (non thermodynamic) computer simulations will not estimate airflow distribution where natural ventilation pressures are significant (i.e. where significant temperature differentials exist, or where the driving pressures in individual parallel circuits are low).
- Computer simulations will not provide an accurate estimation of all airflows in every part of the ventilation circuit. In many mines (particularly larger ones), there are too many variables, which change on a regular basis (e.g. ore passes or stopes being bogged empty), for any real certainty in all predicted airflows.
- Computer simulations will not design your ventilation system for you! Important aspects like the economically optimum shaft sizes etc must be determined by separate calculations.

8.4 Input Data

The old adage of “garbage in, garbage out” well and truly applies with ventilation simulations. Some brief information is provided here on some aspects of input data.

8.4.1 Simulation Model Layout

For many years the use of a “skeleton” model has been seen as the best approach. In these models main airways are included, but minor ones (e.g. small escape rises etc which do not carry significant airflow) are excluded.

There are a number of advantages in this approach including:

- The model is a more manageable size, reducing the chances for data entry error
- Quicker simulation time and faster screen manipulation for 3-D programs
- Minimal loss in accuracy if removal of unimportant airways is done with care

As always there are also a number of disadvantages

- The person who developed it owns the model, and they are generally the only person who understands it.
- Does not allow for any predictive work other than primary fan duty to be undertaken
- Does not allow the model to be used to “problem solve” the ventilation system

Today’s software packages allow the use of a large number (20,000) of branches to be used. This along with the ability to import data from other software used to design mine openings, means that ventilation models can be constructed to represent every opening in the mine. This provides the advantage of other people being able to ‘view’ the network in the same manner they would view survey or design plans.

8.4.2 Number of Airways

Modern simulation programmes have the facility to import centre line string data from 3D mine planning packages. Before importing the data into the simulation package, it is wise to reduce the number of points used to define the model. For example, the raw data may use 20 or more points in series to define a bend in a decline, whereas this could perhaps be reduced to three or four points, whilst still retaining a presentable 3-D format. The advantages are a less unwieldy simulation model with very little reduction in accuracy.

8.4.3 Friction Factors

Friction factors vary for different types of airways (and different airway sizes). If at all possible, gauge and tube pressure surveys should be undertaken to determine “typical” local friction factors.

8.4.4 Shock Losses

Shock losses seem to be a major source of misunderstanding and error in ventilation simulation work. Two common problems are evident:

- One program applies a shock factor to every new airway by default. If the model is constructed from a large number of very short airways in series, the resulting combination of default shock losses can cause the simulation model to substantially overestimate the total mine resistance. Note that the Atkinson Friction Factor for drives, etc includes an allowance for “normal” shock losses (decline bends etc). Shock factors generally only need to be applied to “dog legs”, inlet losses and discharge losses in main ventilation raises etc.
- One program misleads the user into believing that an “equivalent length” shock factor can be used to accurately describe a particular shock loss (e.g. right angle bend), regardless of the airway size or roughness.

The problems caused by the equivalent length concept are almost universal. Some notes outlining the problem in more detail are presented below:

The equation for the equivalent length of a shock loss is given as $Le = \frac{XDh}{6.67k}$

Where: Le = equivalent length of shock loss (m)
 X = shock factor
 k = Atkinson Friction Factor). Ns^2m^{-4}
 Dh = Hydraulic diameter (m) $= (4 * A) / C$
 A = cross-sectional area (m^2)
 C = drive perimeter. (m)

It can be seen from the above equation that the equivalent length is directly proportional to the hydraulic diameter and inversely proportional to the Atkinson friction factor of the equivalent length airway (i.e. equivalent length for a given shock loss is not constant, but depends on airway size and roughness characteristics). For illustration, using the above formulae, the equivalent length of a discharge loss ($X=1$) from a 1.2m airleg rise ($k = 0.015$) = **12m of 1.2m x 1.2m airleg rise**. In contrast, the equivalent length of the same discharge loss ($X=1$) from a 4m diameter raise bored hole ($k = 0.004$) = **150m of 4m raise bore**. That's quite a difference in equivalent length!

It is important to recognise and understand the problem outlined above. Until there are some changes made in how shock losses are dealt with in some of the simulation packages, it is strongly recommended that all shock losses be calculated separately.

“Woods Practical Guide to Fan Engineering” Daly, B. B. (1978) is a very useful source of information on shock factors. (Note that Daly uses ‘K’ for the shock factor, rather than ‘X’)

8.4.5 Fans

Ventilation programs have the facility to enter fan curves. Most programs require fan pressure in terms of **fan static pressure**. Unfortunately, some fan suppliers only provide fan total pressure

curves. A fan static pressure curve (including pressure losses or regains from associated fan inlet and discharge ducting) should always be requested when ordering primary mine fans.

When simulating a new mine to determine the required primary fan duties, the required fan static pressures can be determined by “fixing” the flow in the primary exhaust airway(s). The required fan static pressure can be read off the simulation results.

8.4.6 Fixed Airways

Fixed airways should be used very sparingly. Excessive use of fixed airways will overly constrain the simulation model and may result in errors or program crashes. Fixed airways are generally only used for airways where the resistance is difficult to measure (e.g. lack of accessibility etc). Even then this should be a temporary fix and once the resistance has been determined the fixed airflow should be replaced with the appropriate resistance. Most of the time, the only airways that should be fixed are airways with ventilation controls (e.g. fans or regulators). Fixed airways should never be applied to locations where there are no ventilation controls (e.g. the flow in a decline should never normally be fixed, unless a control such as a set of doors or booster fan is planned).

If a large number of fixed airflows are required, it generally means that airway resistances have been incorrectly determined and therefore need to be adjusted. (i.e. the wall roughness or the cross sectional area needs to be looked at.)

8.5 Results:

For existing mines, “Calibration checks” against existing fan pressures, regulator pressures and major airway flows should always be made, before using simulation models to predict future ventilation system performances. An appreciation of the limitations in accuracy with the technique (due to uncertainties in input values) is required. Agreement to within 15% of actual airflows is about as good as can be expected.

“the best way to learn the practice of mine ventilation is to slug it out in an environmentally tough underground mine for several years. If a person is devoted full time to the study and practice of mine ventilation and has access to another well qualified mine ventilation engineer the average engineer will start to become reasonably proficient in the basics of mine ventilation after a period of one to two years”

F. Bossard - Manual of Mine Ventilation Design Practice

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AMC Consultants Pty Ltd

www.amcconsultants.com.au

MELBOURNE

Level 19
114 William Street
Melbourne Vic 3000
Australia
Telephone: +61 (0)3 9670 8455
Facsimile: +61 (0)3 9670 8311

email:
amcmelb@amcconsultants.com.au

PERTH

9 Havelock Street
West Perth WA 6005
Australia
Telephone: +61 (0)8 9481 6611
Facsimile: +61 (0)8 9481 6622

email:
amcperth@amcconsultants.com.au

BRISBANE

Level 8
135 Wickham Terrace
Brisbane Qld 4000
Australia
Telephone: +61 (0)7 3839 0099
Facsimile: +61 (0)7 3839 0077

email:
amcbris@amcconsultants.com.au

UK

Hampton Business Centre
7 Mount Mew
Hampton-on-Thames
Middlesex TW12 2SH
UK
Phone: +44 (0)20 8213 5881
Fax: +44 (0)20 8979 2187

email:
amcuk@amcconsultants.com.au

